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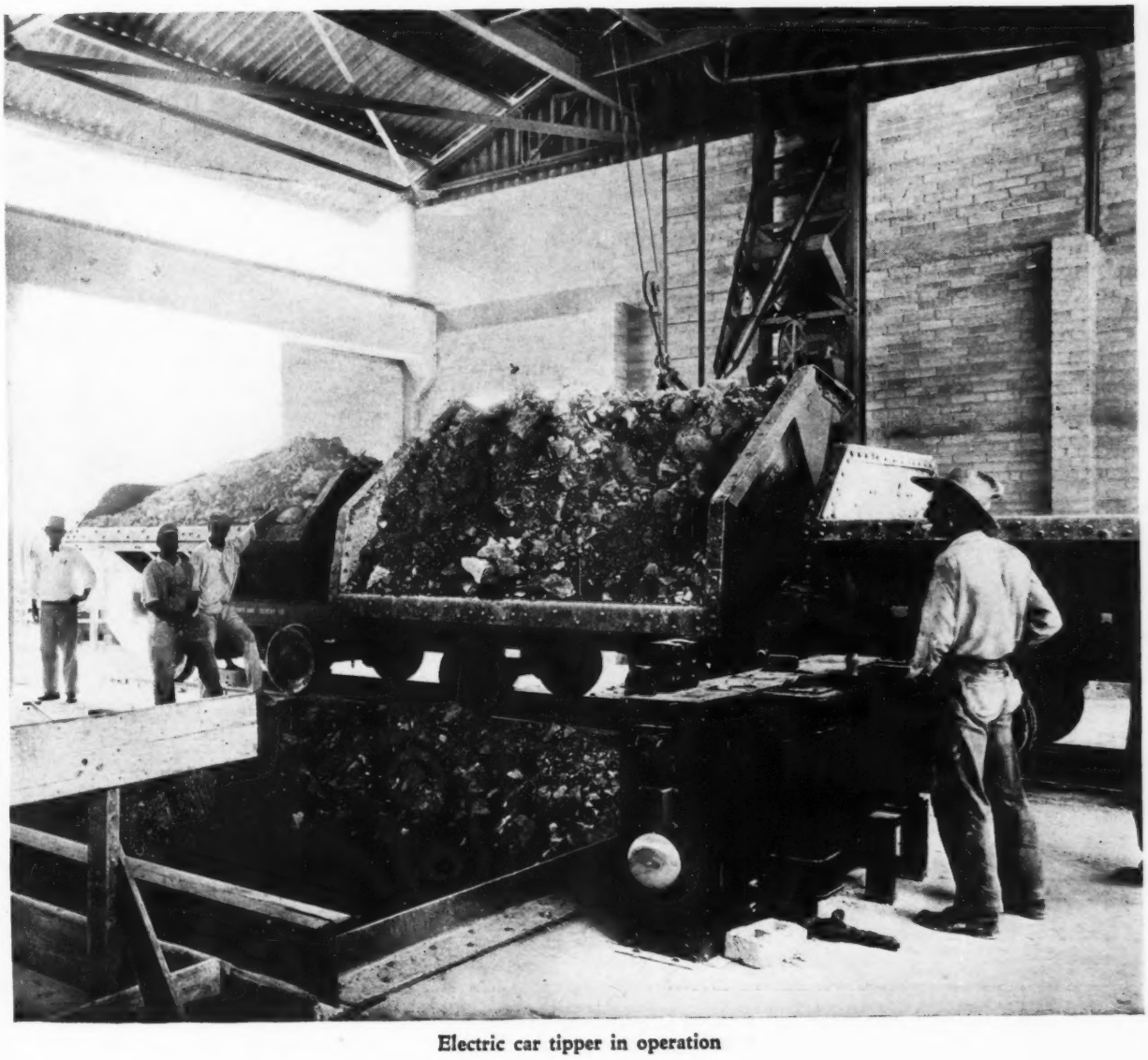

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Engineering and Mining Journal

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APRIL, 1933 VOL. 134—NO. 4

Industrial Competition —Free or Restricted?

Unrestricted competition in the production and sale of the so-called capital goods, such as machinery and supplies used in producing other necessaries, has brought benefit to the consumer of such commodities. Prices are inevitably lower than otherwise they would be—a natural effect of competition and one which at times is accentuated by the overproduction that lack of restriction fosters. Besides enjoying this one-sided advantage, the consumer profits from the great variety of products available and the constant improvement made in their design under the spur of rivalry. Thus he has a better chance to exercise his choice and to supply his needs more exactly when buying.

The inventive process is cumulative. Development of new materials, as, for example, corrosion-resisting metals; the devising of improved tools, such as better drill steel; the application of new principles, as in welding—all these have made possible the seemingly endless stream of new models, types, and styles of machinery and equipment that has come from factories and plants. A typical instance is the recent development of an electric motor, improved by the adoption of stainless steel for the corrugated radiating surface, and introduced to an industrial world seemingly already provided with a motor for every need, yet able to use the new product to advantage.

Restriction of competition has been advocated in various quarters, though not to a degree creating monopoly. Whether or not the advantages of such an innovation (or ancient practice) to society would outweigh its disadvantages is open to speculation, in the absence of experience under present-day conditions. Consumers of plant equipment may wonder whether it would result in a less convenient market in which to buy, and whether the collective ingenuity and inventive skill of the manufacturers in any particular industry are not in proportion to their number. In any case, consumers should remember that, with the honors for the development of mining and metallurgical equipment divided between user and manufacturer as they are, the former is always in a position to give helpful suggestions to the latter, who is always glad to get them.

Restriction of competition might indeed produce a more efficient industrial structure. But inasmuch as the most efficient form is that obtainable only under a monopoly, by its nature undesirable, mere efficiency is obviously not the goal toward which to strive. The interest of the public is twofold: the consumer must not be exploited, nor must the progress of invention be retarded.

Application of Magnetism Finds New Opportunity

Upward lixiviation, for the removal of a soluble salt or element from an insoluble gangue, has been applied in practice for the recovery of non-ferrous metals and some of the so-called non-metallic minerals, particularly nitrate, with a high degree of technical and economic success. Upward clarification, for the removal of suspended matter in water or solution, is equally practicable, but its adoption has lagged since it was first applied several years ago, because of the gradual decrease in permeability caused by the retention of the contaminant in the interstices between the fibers or particles of the substance used for entanglement, even after reverse flow for cleansing. This disadvantage has been overcome in an exceedingly ingenious manner by Mr. W. C. Laughlin, who has carried to a successful conclusion work commenced in Mexico on treating silver ores, and who has applied a new principle to the clarification of liquids that will find application in many chemical and metallurgical operations.

The thickening of solids that settle easily, or of solids that can be coaxed to settle by a coagulating agent, is not difficult; and, during the settling operation, such solids usually carry with them some of the colloidal matter present. The finely divided material that follows the flow of a slowly rising current, however, is less easily separated from the fluid medium. To trap the particles, and to insure continuous operation of an apparatus used for the purpose, with regular flow of clarified effluent through a moderate volume of filter medium, Mr. Laughlin uses a bed of magnetite sand, resting on a false bottom of slotted screening.

In the selection of magnetite for this purpose lies the essence of the invention. The filter is arranged in the form of an annular trough of sand near the upper part of a suitable circular receptacle equipped for the gravitational removal of settled solids from a central point in the bottom, on the Dorr thickener principle; and this sand is momentarily lifted and gently agitated by means of a traveling solenoid, whereby a volume of clarifying material represented by an area of twelve to fifteen square feet is continuously cleaned per minute. The solid matter so released by the upward rush of liquid through a part of the bed thus made to offer a lesser resistance to flow is carried away, for return to the main circuit or to be discharged, by means of a motor-operated centrifugal pump that travels with the solenoid. Efficient clarification is therefore effected without recourse to a pressure beyond that supplied by a hydrostatic head of a few inches of liquid.

The Laughlin development should be of interest to

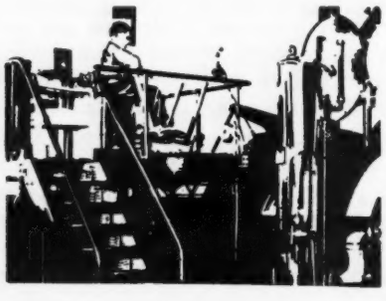
readers of *Engineering and Mining Journal* on at least two counts. It indicates the scope for the successful utilization in other fields of the versatility of mining technicians in applied chemistry, metallurgy, ore dressing, physics, and mechanics; and the advantages that follow a broad rather than a highly specialized experience. Mr. Laughlin's filter has found large-scale application in the treatment of sewage water; but it promises to solve many problems that beset the hydrometallurgist, especially in regard to the clarification of corrosive liquids. The invention, like many another, illustrates the opportunity that exists for mental and experimental research by engineers with practical experience in the needs of industry and the properties of inanimate matter.

Early in 1913 the writer of these notes devised and operated in Africa an upward-flow sand clarifier for removing suspended matter from cyanide solutions, subsequently elaborated as a zinc-box clarifier and described in *Mining and Scientific Press*. Cleansing of the filter medium was effected periodically by stopping the inflow and opening a sludge exit at the bottom of the apparatus, which caused a reverse flow that freed the filter of most of the accumulated solids. The Laughlin invention avoids the disadvantages under which this clarifier operated. It provides for continuity in operation, freedom from the need for personal attention, and effective cleansing of the filter bed, which insures high capacity per unit of sand volume or surface area. Inquisitive technicians may be expected to seek new opportunities to utilize magnetite under solenoid control for purposes other than the clarification of liquids.

Finding New Outlets For Mine Products

In a recent issue of *Engineering and Mining Journal* was reported a substantial increase in the output of salt in 1923 from the Malagash mine, in Nova Scotia; and mention was made of the fact that the company owning and operating the property was developing a new outlet for its low-grade product, in conjunction with local clay, as a road-surfacing material. The pioneer in this development is Mr. A. R. Chambers, of New Glasgow, N. S., who succeeded in interesting the Nova Scotia Department of Mines and the Department of Mines of the Dominion, with the result that an investigation is being made at Ottawa to determine the precise nature of the reactions involved.

Although the clays used in the Nova Scotia tests have proved highly satisfactory, the results cannot be duplicated with all types of clay. High plasticity is apparently needed to insure the production of a surface layer that will withstand disintegration under heavy traffic. The method of procedure on an eighteen-foot road in Nova Scotia is to spread the clay—about fifteen tons to the mile—on a gravel foundation. Salt is then added in similar amount, and the mixture wetted and rolled, forming a plastic mass that appears to lose water at the surface without involving a drying effect that would lead to dusting. Repairs, when necessary, are easily effected with a mixture of equal quantities of clay and salt.



Although the innovation has not been under trial long enough to provide conclusive evidence as to the permanence of such a surfacing, the facts disclosed indicate that a new use for salt and clay, especially in outlying districts, has been developed. The experience in Nova Scotia illustrates that initiative in seeking new outlets for mine products and byproducts may prove of substantial help in lowering cost per ton by permitting increased output. It demonstrates anew that success in exploiting a natural deposit is to be found in research and experimentation, and the application of broad knowlege and practical experience. Too many owners of a minable mineral are inclined to formulate their sales plans after an answer to the inquiry "What is it being used for?" instead of asking themselves "To what new use could it be put?"

Engineers and the Public Need Protection

The latest lure for the unwary investor in gold mines comes to him in the form of a "Registered Gold Credit Check" for $90, inclosed with a florid appeal from a promoter who reports that he has secured a group of "famed" gold-mining properties in the Western United States. "The gold ore is being taken out of the mine now," the prospective victim is told. "A picked crew of miners are at work, and the force is being increased still greater, right now, to take out more gold ore—to step up the mines out-put of the precious stuff. There is," one is assured in all seriousness, "positively no doubt or question about gold ore being here. . . . The ore opened up, in reserve, or in sight," according to a "noted mining engineer" whose name is not to be found in "Who's Who in Engineering" or in the directories of the professional mining and metallurgical societies, "together with his estimate of probable ore, and physical assets, as per his report, runs into the astounding sum of more than nine million dollars."

"I have chosen you," the invitation reads, "as an outstanding citizen of your community. I want you, as such, for my associate. Therefore I am giving you this $90.00 Registered Gold Credit Check, which pays for 90 shares in the ——— ——— Gold Mines stock; and to bring the total up to 100 shares—$100.00 worth at par value—you remit the paltry sum of only $10, payable in this utterly astounding manner: $5 with your acceptance of the Registered Gold Credit Check for $90.00, and not another cent until I announce to you your first cash payoff, whereupon you will remit $5 more, making a total of $10 in all which you will remit, for which you get one hundred shares of $1 par value, making a hundred dollars worth of stock in this—which I consider destined to be one of the greatest gold mining properties of a similar state of development in all the entire United States."

The appeal would be amusing if it were not so pathetic an indication that money is available to be squandered on such an unsubstantial venture. As this publication has insisted in many previous issues, the blame cannot be attributed wholly to the stupidity of the investing public.

While this editorial was being written a request was received for information as to the integrity and ability of ten "mining engineers" and "engineers of mines" who had reported on gold properties in which money had been invested by our correspondent, the shares in which had apparently been offered as "an attractive speculation." Only one of the names could be found in the directory of the A.I.M.E., and none in the membership list of the Mining & Metallurgical Society of America.

So long as promoters are permitted to bolster their appeals for cash with the alleged findings and opinions of an "engineer" lacking an easily ascertainable technical record or professional affiliation, without supplying evidence that he possesses qualifications of training and experience; so long as the writer of a prospectus can elevate anyone to the high standing of a "noted mining engineer," and advertise the pretension through the medium of the United States mails, as is being done by the distributor of the "Registered Gold Credit Check," the public will continue to be fooled and fleeced.

Metallurgical Practice at a Famous Gold Mine

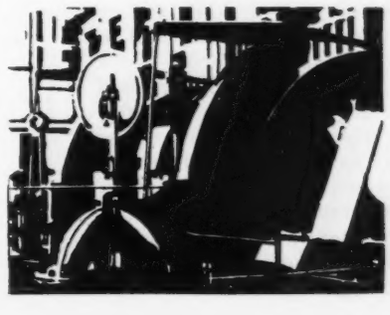
Metallurgists throughout the world have waited long for the details of plant practice at the mill at Morro Velho, Brazil, of the St. John del Rey Mining Company, operating one of the deepest gold mines in the world. These data were recently disclosed in a monograph presented at a meeting of the Institution of Mining & Metallurgy, by Messrs. J. H. French and H. Jones, which created unusual interest and prompted lively discussion. The authors prefaced their paper by a description of the ore that included results of microscopic examination by Mr. T. Haden, which indicated that the gold is generally associated with arsenopyrite and in contact with pyrrhotite. The importance of microscopic analysis as an aid in the development of a correct metallurgical technique is not widely recognized. Of interest to note is the fact that the association of the gold with arsenopyrite and in contact with pyrrhotite is similar to an occurrence at the Homestake mine, in South Dakota, reported in 1924 by Mr. Sidney Paige, of the United States Geological Survey.

Pyrrhotite is an easily oxidized mineral that sometimes escapes detection by cyanide metallurgists who are at a loss to explain high solvent consumption and contaminated solutions. The authors mention that "References to the action of pyrrhotite on cyanide solutions are very rare in any of the standard textbooks on cyanide metallurgy." True, but the explanation is simple. Textbooks consist primarily of fundamental theory and established practice. The latter must be drawn from published articles and monographs and criticisms in the technical press and in the transactions of professional societies. Dearth of textbook information on the action of pyrrhotite on cyanide solutions is the result of the hesitancy of those who for many years have been in close touch with the problem to share their experiences with their confrères, to describe plant practice, to record metallurgical result. The French-Jones paper and the discussions that follow it will provide the information needed and be welcomed by writers of textbooks.

Practice at Morro Velho, publication of the details of which will prove especially helpful to the operators of plants treating a pyrrhotite ore, involves stamp milling to 50 or 60 mesh and inexpensive concentration on canvas strakes and canvas tables to a 50 per cent gold product, suitable for direct melting. Cyanide practice is complicated by the destructive action of the ground pyrrhotite on the solutions, and the apparent difficulty experienced in the control of an important factor—alkalinity. Precipitation is effected by the use of filament zinc, made on the property from ingot zinc, to which is added a small amount of lead. The tailing solution from the precipitation boxes, carrying nearly a kilogram of cyanide per ton treated, is discharged to waste.

Discussion of the paper disclosed several interesting suggestions. More than one speaker opined that economies might be effected by the acquisition and operation of a cyanide regeneration plant. Mr. R. J. Lemmon and Mr. B. L. Gardiner added to common knowledge on the rôle of pyrrhotite in cyanide practice by a description of probable reactions, illustrated by means of those essential aids to understanding—chemical equations. Both recommended an earlier control of alkalinity than is the practice at Morro Velho, and pointed out that the absence of protective alkalinity in the solutions entering the precipitation boxes was a contributory cause of the plating of the zinc with gold, instead of the production of a more desirable loose precipitate. Mr. Harley B. Wright essayed to predict the result of adopting flotation, discarding concentrator tailing, and roasting the concentrate prior to cyaniding it. "Precipitation troubles would vanish," he added, "and altogether the picture would be one of happiness and contentment for the metallurgist." This simple alternative to the complicated metallurgical system in vogue at Morro Velho, the unavoidable result of many decades of continuous operation, appears attractive and economical; but fundamental changes in practice are not effected with indecent haste on a property that has been a steady producer for a century, with a mill that has seen service for nearly forty years.

The St. John del Rey company has been criticized in the past because of its policy of secretiveness in regard to metallurgical operations. It is now to be congratulated on a change in attitude that brings it into metallurgical fellowship with all but a very few of the great gold mines of the world. It has a fine record for continuous and profitable operation, despite the refractoriness of its ore, over a period that far exceeds the life span of most gold mines. Its metallurgists have followed their own designs and developed their own processes, uninfluenced by changes in metallurgical fashion elsewhere. The result has been an uninterrupted employment of a staff of engineers and technicians, of skilled and unskilled laborers, and a satisfactory return to stockholders without liquidation of large company investments in British Government securities. In this respect the directors and executives of the company have fully met the primary obligations involved in the exploitation of a great natural resource. They are in a particularly favorable position to meet necessary expenditures that may arise in the future for plant modernization.



Smelting and Refining Technique at Port Pirie, Australia

O. H. Woodward

General Superintendent, Broken Hill Associated Smelters Pty., Ltd., Port Pirie, South Australia

THE greater part of the lead ores treated at Port Pirie, Australia, is derived from the mines at Broken Hill. In addition, small parcels of concentrate, as well as sulphide and siliceous lead-bearing ores, are received from various mines scattered throughout the Commonwealth. Material treated is divided into three main groups: (1) granular concentrate, the product of jigs and Wilfley tables; (2) slime concentrate, the product of flotation units; (3) siliceous ores, received in the crude state. These are crushed at the plant so that the bulk of the material passes through a $\frac{1}{8}$-in. mesh screen. The plant also treats Risdon residues—the designation applied to the residues obtained from the plant of the Electrolytic Zinc Company of Australasia, Ltd., situated at Risdon, Tasmania.

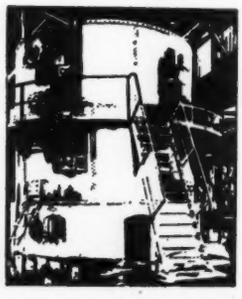
A typical roasting and sintering charge has the following composition: concentrate, 25 per cent; slime, 32 per cent; siliceous ore, 2 per cent; lime-sand, 3 per cent; Risdon residues, 19 per cent; granulated slag, 19 per cent. When siliceous ore is not available the silica is supplied by use of siliceous sand. Besides these components, pre-roast, to the extent of 60 to 100 per cent of the total of the other constituents, is added to keep the charge opened up. The lime-sand flux is a disintegrated wind-blown limestone sand obtained on Wardang Island, Spencer's Gulf, South Australia. A typical analysis of the sand is as follows: SiO_2, 5.5 per cent; FeO, 0.17 per cent; CaO, 48.30 per cent, or 86.25 per cent $CaCO_3$; MgO, 3.00 per cent, or 6.30 per cent $MgCO_3$; Al_2O_3, 0.60 per cent.

The various components of the charge are delivered to a series of bins in the mixing plant, where regularity of delivery of the respective charge components to the main charge-collecting conveyor insures even mixing. In design, each of the bins is a truncated cone, with the bottom section in the form of a frustrum of a pyramid, so that on four corners an opening exists through which an attendant can operate to prevent the material hanging up. A 36-in. rubber conveyor belt runs under the bottom of the bin, and immediately in front of the bin door is placed an extension, beyond which is the regulating door. By this means, even though the material in the main bin may hang up, a sufficient supply of free-running material is available to prevent interruption in delivery. Little trouble is experienced in keeping a regular discharge of the material to the main charge-collecting conveyor.

The conveyors under the respective bins deliver the charge components to a main charge-collecting conveyor running transversely to the main series of bins. The charge is then continuously delivered to the supply hoppers of the Dwight-Lloyd units.

Preliminary roasting is carried out on seven straight-line, standard Dwight-Lloyd machines. The conveyors delivering the charge from the supply hoppers to the machines are direct geared to the machines, and deliver the charge to the pallets by means of oscillating chutes working transversely. At the point where the conveyor delivers the charge to the oscillating chute the correct moisture is added. The oscillating chute delivers the charge evenly across the width of the pallet, the stream first falling onto an inclined deflecting plate that takes the impact of the falling charge, effectively counteracting the tendency to form a densely packed charge on the pallet. Again, as the charge rills off the inclined deflecting plate, the larger nodules roll down to the toe of the rill, and thus form a uniform bed on the grate, preventing clogging.

The charge, having been evenly and lightly distributed over the pallets, passes under an adjusting leveling plate, by which the depth of the charge is regulated. After passing the leveling plate, the charge passes under the ignition stove. This stove is merely a firebox with an undergrate blast, the products of combustion being delivered to the surface of the ore charge by a downcast flue. A coke fire is used in the stove to obtain a clear gas, preventing the surface of the charge being covered with carbon, as it would be if a high-hydrocarbon coal were used. Operating data for each of the Dwight-Lloyd machines is presented in Table I.



After the charge passes over the length of the machine, it is automatically tipped into a hopper at the end of the machine, and discharged by a screw onto a collecting conveyor, which delivers it to the intermediate crushing section.

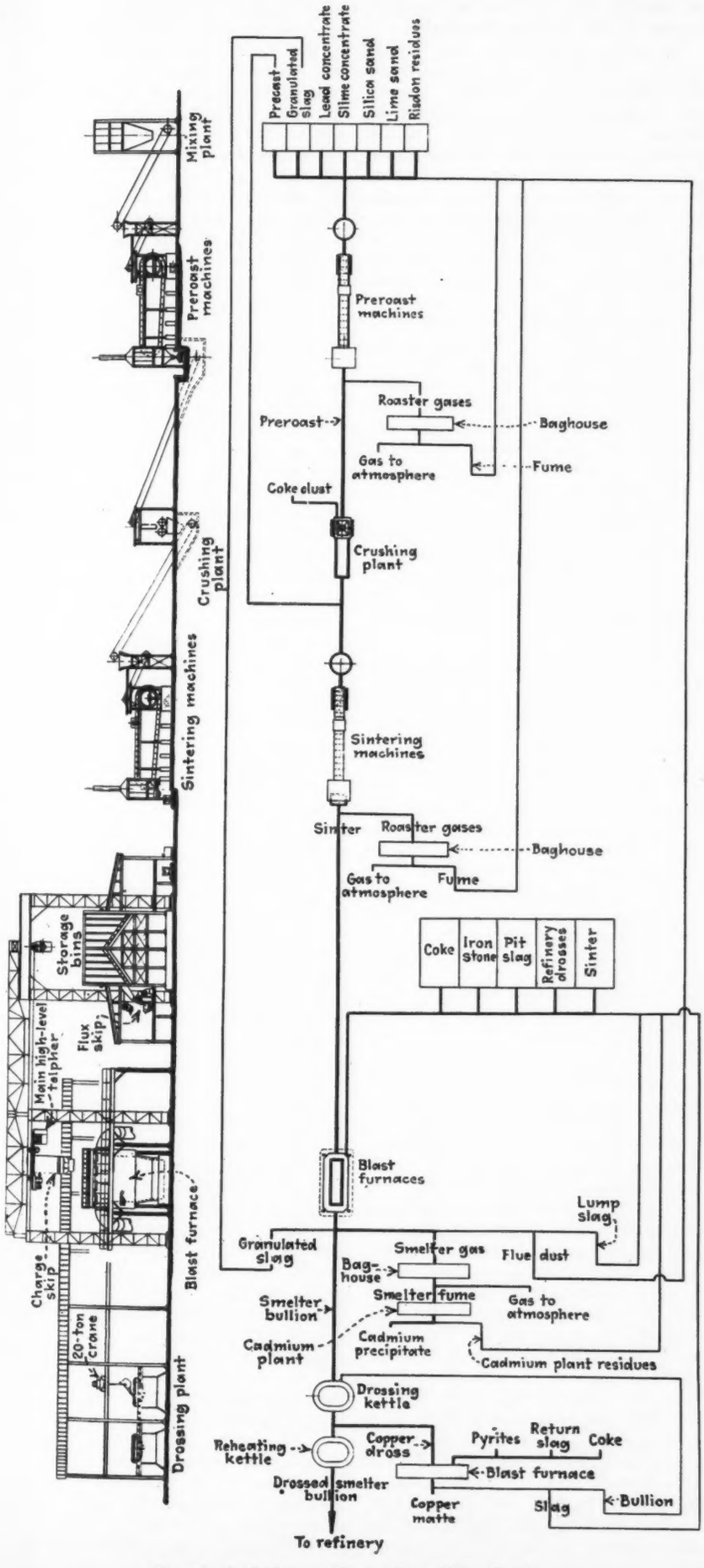
Sulphur content of the preroasted charge averages around 5 per cent, and the greater part of such sulphur is found as unaltered sulphide that forms the kernel of individual lumps of sinter. Consequently, to bring these kernels into a physical state to permit the removal of the sulphur in the final roasting, the preroasted charge is crushed so that about 60 per cent of the crushed material will pass through $\frac{1}{4}$-in. mesh. Crushing is carried out in smooth-faced crushing rolls.

Table I—Operating Data for Each Roasting Machine

Moisture of charge, per cent	6
Thickness of charge layer, in.	3½ to 4
Average speed of machine, in. per min.	33
Vacuum in chambers, water gage, inches	8
Volume of air, cu.ft. per min.	16,000
Coke used for ignition, lb. per ton of charge treated	13.2
Charge treated per machine-hour, tons	9.24
New input lead treated per machine-hour, tons	4.15

The final stage of roasting or sintering is carried out on six Dwight-Lloyd machines, similar in design to the seven preroasting machines. Depth of the layer of charge on these machines is 8 to 8½ in., as compared with 3½ to 4 in. on the preroasting machines. Grate travel averages about 12 in. per minute. Typical analysis of the sinter is: lead, 45.90 per cent; silver, 16.70 oz. per ton; sulphur, 1.90 per cent; zinc, 10.23 per cent. Each final sintering machine is equipped with a small storage hopper having a self-closing door. This door is operated by a solenoid, which, in turn, is operated by a tripping gear attached to the main sprocket wheel of the machine.

The relatively hot pallets of sinter discharged from the machine would tend to pack were they stored in the hopper for any length of time. Ten pallets of sinter normally constitute the individual charge to the blast furnaces, and, as the weight of a cubic foot of charge to the roasting machines is taken twice per shift, and as the charge on each pallet is accurately leveled off, weighing of the sinter portion of the blast furnace charge is dispensed with. In starting up a machine, a standard telpher skip is placed under the hopper, the door of which is opened. Each pallet of sinter, as it is discharged from the machine, falls through the bin into the telpher



Flowsheet of the roasting and smelting plants

skip. When ten pallets have passed through, the control on the main sprocket wheel of the machine brings the solenoid into action, which, in turn, trips the door of the hopper so that the door closes automatically. Any pallets of sinter discharged from the machine during the interval between withdrawal of the loaded telpher skip and its replacement by an empty skip are stored in the hopper. When the new skip is in position, the door of the hopper is opened by hand and the same procedure repeated.

The telpher skip with its charge of ten pallets of sinter is withdrawn by means of a storage-battery truck, and, depending on circumstances, is delivered to either the blast furnaces or the intermediate storage bin. If the charge of sinter goes direct to the blast furnaces, it is delivered to the secondary telpher system, and the necessary part of the blast furnace charge is added. The skip is then moved to the main telpher system, and transported to the blast furnaces. Sinter not immediately required by the blast furnaces is delivered to the main telpher system, and transferred to the storage bin.

Gases from the preliminary and final roasting units are delivered to separate baghouses of standard design. Owing to the high temperature of the gases (255 to 265 deg. F.) leaving the machine fans, a Green's economizer has been installed with each baghouse. The standard bag is made of flannel, with a weave 22x22 in warp and weft. Each baghouse is equipped with an exhaust fan, by which the filtered gases are discharged to the atmosphere through a high stack.

In the baghouses of the preliminary roasting section, the average temperatures of entering gases is 170 deg. F.; maximum temperature allowable in entering gases is 190 deg. F.; quantity of gas filtered per minute per square foot of bag area is 0.75 cu.ft.; fume caught per square foot of bag area per minute is 0.00007 lb.; the pressure drop across bags averages 2 in. of water gage; and the maximum pressure drop is 2.5 in. of water gage. In the baghouses of the final roasting section, gas temperature is the same as that in the preliminary roasting section; the quantity of gas filtered per minute per square foot of bag area is 0.8 cu.ft.; fume caught per minute per square foot of bag area is 0.00003 lb.; pressure drop across bags averages 2 in. of water gage; and maximum pressure drop is 2.5 in. of water gage. All fume caught is passed through a paddle-blade mixer, nodulized, and returned to the preliminary roasting section of the plant.

A typical blast-furnace charge consists of: sinter, 6,600 lb.; returned slag, 800 lb.; ironstone, nil to 100 lb.; coke, 660 to 760 lb. Returned slag is that quarried from a slag dump, and the prime factor in its use is to dilute the zinc content of the slag eventually pro-

duced. Should normal sinter be smelted, the slag which would eventually be produced would contain 22.52 per cent zinc or 28.15 per cent zinc oxide. Although the furnaces could be run to produce a slag carrying 22.52 per cent zinc, Port Pirie practice has been to consider a slag carrying around 18 per cent zinc as most economical from the smelting point of view. Therefore, to reduce the zinc in slag eventually produced to around 18 per cent, slag quarried from the old slag dump is added, as such slag is relatively low in zinc.

The telpher installation, placed in full operation on Jan. 17, 1929, was installed to facilitate assembly of blast-furnace charge materials, as well as the preparation and eventual delivery of the completed charge to the blast furnaces. [The installation was described fully in an illustrated article published in *Engineering and Mining Journal* of Oct. 26, 1931, pp. 356-58.] It consists of three sections.

No. 1 section: 395 ft. long, running north and south, parallel to the water front, and so located that coke discharged from ships can be delivered by 7½-ton Babcock-Wilcox luffing jib cranes to a stockpile. Coke reclaimed from the stockpile for use in blast furnaces is first screened in a central screening plant, and then delivered to the coke bin in the main storage bin system.

No. 2 section: 456 ft. long, running east and west, and connected on its eastern end with No. 1 section, and on its western end with No. 3 section. This section spans the main storage bin system and the operating yard, in front of the Dwight-Lloyd blast roasting machines.

No. 3 section: 386 ft. long, running north and south, and spanning the blast furnaces.

The telpher tracks, upon which the telpher trolleys travel, consist of runway girders or monorails suspended from the main structure at a height of 58 ft. above ground level. Telpher trolleys are for 5-ton lift, and are electrically operated on 550-volt 3-phase 50-cycle alternating current. A trolley, with its load of 5 tons, weighs 11 tons 15 cwt., and travels at a speed of 400 ft. per minute. An area of two acres is covered by the telpher installation.

The main storage bins have a total capacity of 2,630 tons, as follows: sinter, 1,200 tons; coke, 170 tons; slag, 430 tons; ironstone, 230 tons; copper dross, 300 tons; lead dross, 300 tons. The bins discharge on either side, and the respective materials can be discharged into the skips of the secondary telpher installation. Secondary telpher trolleys carry a 2-ton load, and are fitted with traveling gear only. By the aid of the secondary telpher, the respective components of the blast-furnace charge are collected and eventually delivered to the standard telpher skip containing the sinter portion of the charge. The completed blast-furnace charge is eventually delivered to a super hopper above each blast furnace. This hopper discharges on either side, and the charge to the furnaces is regulated by discharge gates.

The smelting plant consists of six large furnaces and one small furnace; the latter has a capacity about two-thirds that of a large furnace. The number of furnaces in operation depends on the rate concentrate is received from the various Broken Hill mining companies.

Bullion, discharged almost continuously from each blast furnace through one of two siphons, flows into ladles of 5 tons' capacity, which are then picked up by a traveling crane and transported to the drossing section. Slag is tapped every 30 min. from either end of the furnace, but usually from one end only, into a 1-ton settler. Overflow from the settler falls onto a stream of high-pressure salt water, and is granulated. The granulating launders deliver the slag to the boot of an elevator, and from there it is delivered to storage bins, which discharge direct into railway trucks for transport to the dump. Ordinary rubber conveyor belting is used in these elevators; it gives greater satisfaction than link belts.

The normal blast-furnace slag carries about 22 per cent zinc oxide, and, at times, up to 33 per cent zinc oxide has been carried without serious trouble being experienced in furnace conditions. These results are practicable only with good blast roasting, which results in the oxidation of the zinc sulphide contained in the charge. The slag is perfectly fluid and gives no trouble at the taphole; the tuyeres remain bright and a good separation is effected in the crucible. No surface matte is produced, but the necessity for top-barring the furnace is more frequent the higher the zinc oxide content of the slag. A typical analysis of slag is: SiO_2, 19.8 per cent; FeO+MnO, 34.2 per cent; CaO+MgO, 12.9 per cent; Al_2O_3, 5.8 per cent; ZnO, 21.8 per cent or 17.5 per cent Zn; S, 2.3 per cent. Specific gravity of the slag is 3.66; specific heat, 0.1767; temperature at taphole, 1,000 to 1,200 deg. C.

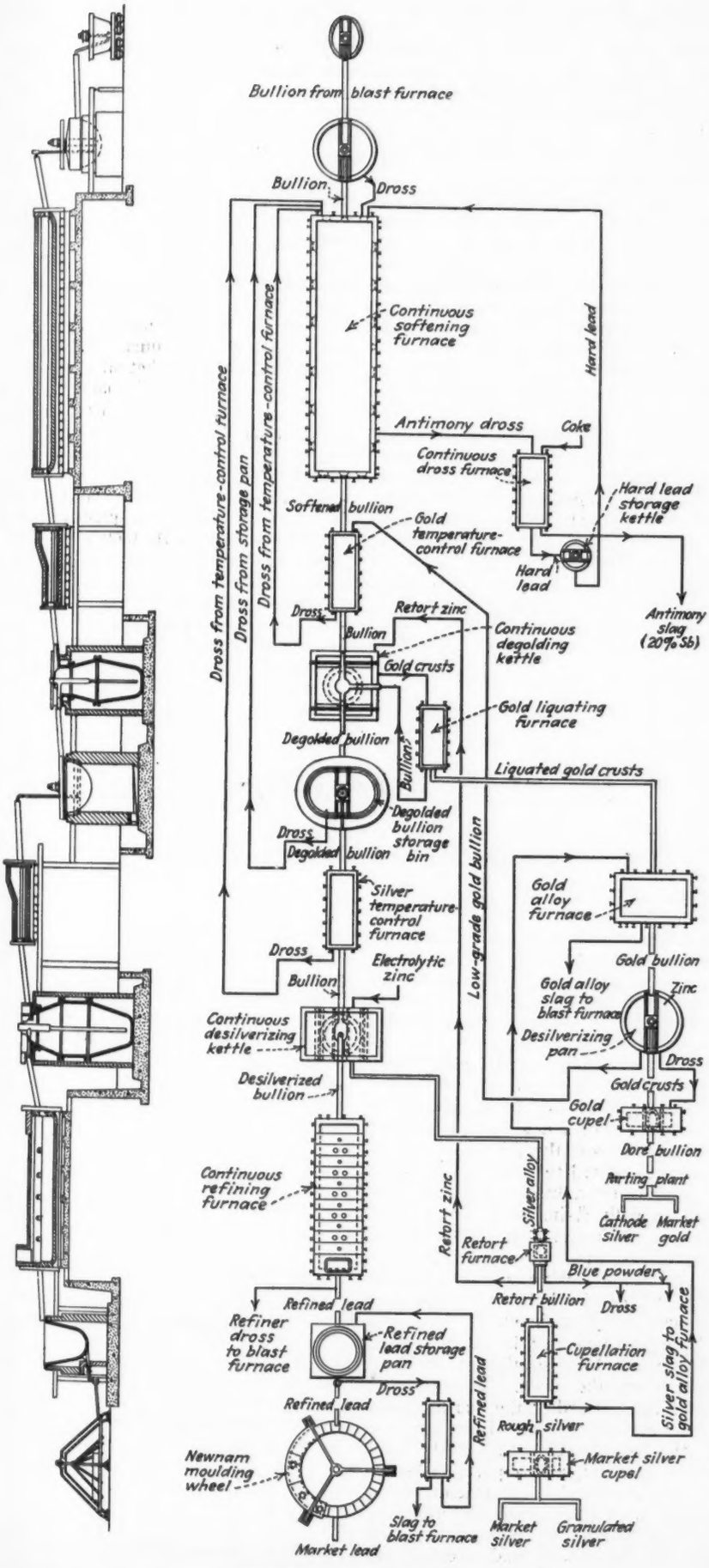
Blast-furnace gases are delivered to a baghouse of standard design, but, in this instance, no economizer is installed to cool the gas. Average temperature of gas entering the baghouse is 170 deg. F.; maximum temperature allowable in gas in the baghouse is 190 deg. F.; quantity of gas filtered per minute per square foot of bag area is 0.87 cu.ft.; fume caught per minute per square foot of bag area is 0.00017 lb.; maximum pressure drop across bags is 2.5 in. of water gage; and the average pressure drop is 2.0 in. of water gage.

Fume collected by the baghouse is discharged to a central sump, agitated with water, and pumped to a storage vat. The pulp is then passed through a vacuum filter giving two products: (1) residue, which is returned to the blast furnaces; and (2) filtrate, which is delivered to another vat. To the filtrate is then added sodium carbonate, which precipitates the cadmium as cadmium carbonate. The cadmium carbonate is removed from the solution by filtration, dried, and sent to the works of the Electrolytic Zinc Company, at Risdon, where it is eventually refined to the grade of electrolytic cadmium. The filtrate obtained after filtration of cadmium carbonate is of no value.

In the drossing section, the copper, which is present in the base bullion as dissolved sulphide, is removed by liquation, the copper content decreasing from 1.5 to 0.06 per cent. The plant consists of five 100-ton kettles, into which the bullion from the 5-ton ladles is poured. The bullion is allowed to cool back to about 350 deg. C., and copper crusts are removed by means of a grid, which is kept below the surface of the cooling bullion. These crusts are treated in one of the blast furnaces, and worked up to lead-copper matte, which is sold. After removal of the copper dross crust, the bullion is pumped to a 15-ton ladle and transported in a molten condition to the lead refinery.

Lead bullion is carried to the refinery in 15-ton ladles, and then pumped to one of two oil-fired bullion storage kettles. Analysis of the bullion as received is: Copper, 0.068 per cent; antimony, 0.410 per cent; arsenic, 0.182 per cent; silver, 47,200 oz. per ton; gold, 0.104 oz. per ton. Bullion is pumped at a regular rate from the storage kettle, and passes continuously through the various stages of refining to produce market lead. The processes used in the refinery are unique in that they are all operated continuously; they have been developed at Port Pirie by the metallurgists of Broken Hill Associated Smelters, all of whom are graduates of Australian schools of mines and universities. These methods have been patented throughout the lead-producing countries of the world.

In the section devoted to refining for arsenic and antimony, two reverberatory furnaces are available; only one, however, is operated, the second acting as a standby. The bullion is heated to about 750 deg. C. by oil firing, the antimony and arsenic being removed by oxi-



Flowsheet of the refinery

dation. Oxides of these metals, together with litharge, are run off to a small reverberatory furnace, wherein coal or coke is used as a reducing agent to produce an enriched antimonial slag and hard lead. The lead, containing about 1.5 per cent antimony, is returned to the large furnace. The slag, which contains about 20 per cent antimony and 56 per cent lead, is delivered to storage bins and eventually smelted in a blast furnace for the production of antimonial metal.

Softened bullion, containing about 0.03 per cent antimony and less than 0.0005 per cent arsenic, gravitates from the lead overflow of the furnace direct to the first unit of the continuous degolding section. The softening furnace consists of a steel tank, 52 ft. 4 in. long, 13 ft. 5 in. wide, and 22½ in. deep, lined with concrete, on which a 9-in. firebrick invert is built. To prevent attack on the brickwork by litharge, water-cooled jackets surround the bath of metal at the level of the slag. The arch, which has a rise of 8 in., is set 2 ft. above the jackets.

In the refining for gold and silver, the softened bullion, at a temperature of 750 deg. C., passes through a small furnace, the function of which is to control the temperature of the exit lead at about 650 deg. C. It then flows to one of the two degolding kettles. Each kettle, 18 ft. 3 in. deep, consists of three circular cast-iron sections. The bottom section is 6 ft. internal diameter at its lowest point, increasing gradually to 8 ft. 2 in. at the flange, and having a height of 9 ft. 2 in. The middle section is in the form of a truncated cone, 5 ft. high, which is set on the bottom section and has an internal diameter of 6 ft. 2 in. at its upper flange. The top section is approximately hemispherical in shape, with the necessary openings for bullion inflow, outflow of degolded bullion, introduction of zinc, and removal of alloy. The various sections are bolted together, and leakage is prevented by a special form of arc-welded flexible joint.

Bullion enters the top of the kettle, where zinc is also introduced. Degolded bullion flows from a cast-iron siphon pipe, which reaches to the bottom of the kettle. The gold-zinc-lead alloy formed in the upper zones settles out of solution as the bullion flows down the kettle, as a result of decreasing temperatures, and it rises to the surface, owing to its comparative lightness. Degolded bullion flows from the siphon pipe at a temperature slightly above the melting point of lead. It then enters a storage kettle, where its temperature is raised before it is pumped to the desilverizing section. It has the following approximate analysis: copper, 0.0098 per cent; zinc, 0.16 per cent; gold, 0.35 grains per ton; silver, 45.70 oz. per ton. At intervals, the alloy is dipped from the degolding kettle and placed in a small reverberatory furnace. It is then liquated at a temperature of 450 to 500 deg. C.,

to form a gold crust and lead of low gold content. The lead is returned to the degolding kettle, and the gold stored for periodical recovery of the gold. The gold crust contains about 15 oz. gold per ton, 350 oz. silver per ton, and 10 per cent copper.

Continuous refining for silver is carried out with three cast-iron desilverizing kettles, similar in design to the continuous degolding kettles, except that the maximum internal diameter, 10 ft., is greater by about 2 ft., and the height is 23 ft. 6 in. This greater height is attained by the introduction of a cylindrical section 5 ft. high, set between the bottom and the conical castings. Normally, only one of the kettles is needed to treat the total production, and the process generally is similar to that of continuous degolding. The alloy is dipped, and then cast in small ingots, which are sent direct to the silver yard for the production of market silver.

Desilverized bullion, containing about 0.56 per cent zinc and 0.03 oz. of silver per ton, flows direct from the siphon pipe to the final refining process, which removes the residual zinc and the last traces of antimony. The silver alloy has the following approximate analysis: silver, 6,000 oz. per ton; copper, 0.5 per cent; zinc, 64 per cent; lead, 15 per cent.

Total zinc consumption is about 21 lb. per ton of market lead. The degolding section uses 8 lb. per ton, and this zinc is generally that recovered from the retorting of silver alloy, whereas virgin zinc is used in the continuous desilverizing process.

Refining for zinc and last traces of antimony is carried on in a continuous refiner furnace similar to the continuous softening furnace, except that it is much shorter, being 31 ft. 9 in. long, as compared with 52 ft. 4 in. Zinc remaining after desilverizing and the last traces of antimony are removed by oxidation. Oil burners raise the temperature of the metal to about 750 deg. C., and the drosses, which float on the surface, are removed in a solid condition. The drosses are returned to the blast furnaces for the extraction of their lead content. Input lead flows continuously to the refiner from the desilverizing section, and gravitates from the discharge of the furnace to the small storage kettle in the molding section.

Market lead is molded by one of two Newnam mechanical molding wheels, the bars being removed from the wheel in sets of five. When eight tiers have been set in position, the stack, consisting of 40 bars, is picked up by an electric truck. The stack is then placed on the platform of a weighbridge, which indicates the stack weight direct, thus obviating the necessity for taring and precluding any possibility of error. Weight of a bar is 89.6 lb., and that of a stack of 40 bars is 1.6 tons.

Annual output of market lead and silver depends on the quantity of concentrate received from the various Broken Hill mining companies. The present plant is capable of producing about 195,000 tons of market lead per annum. For the last three years the average annual production has been 154,900 tons. For the same period the production of market silver has averaged 7,920,200 oz. per year.

In treating byproducts, the copper is removed from the bullion in the drossing section of the blast-furnace department. The crusts, which consist mainly of copper sulphide and entangled lead, are stored for periodical treatment in one of the blast furnaces. From this process the lead bullion is incorporated with that produced from the smelting of sinter, and the copper, recovered as a matte, is marketed.

Gold crusts, recovered from the continuous degolding section, are heated with litharge in a small reverberatory furnace to a temperature of 1,000 deg. C. Oxidized products form a slag known as gold-alloy slag, which is returned to the blast furnaces. The bullion is molded roughly and sent to a kettle, wherein a rich gold crust is obtained by the batch method of the Parkes process. This crust is retorted and cupeled to produce doré bullion, which is electrolytically parted.

The alloy removed from the continuous desilverizing kettles is treated in standard-type, oil-fired, tilting retort furnaces for the recovery of zinc. At the temperature of the retort, about 1,400 deg. C., zinc is volatilized and passes to a condenser. The temperature in the condenser is much lower than that of the retort, reducing the zinc to the liquid condition, in which form it is then tapped off into molds for return to the continuous degolding section. Bullion remaining in the retort after the zinc has been driven off is molded into rough bars, which are subsequently cupeled for the production of market silver.

In the cupellation furnaces, of the small oil-fired reverberatory type, the lead is removed by oxidation, the process being hastened by blowing low-pressure air over the surface of the molten bullion. Molten litharge is run off into pots, and is subsequently treated for the recovery of its metallic content. When all lead has been oxidized, the molten silver is dipped into bars of about 1,175 weight troy ounces. These bars are then dressed and stamped for sale as market silver.

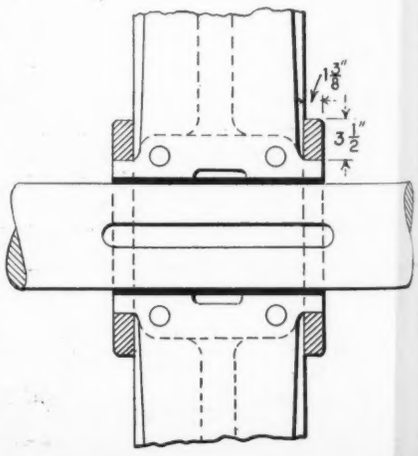
■ ■ ■

Repairing Herringbone Gear of Motor-Driven Hoist

A LARGE HERRINGBONE GEAR on the drum shaft of a motor-driven hoist at the property of the Inspiration Copper Company developed movement with respect to the shaft that prevented keeping the gear true or tight. Dimensions of the gear were: diameter, 11 ft.; face, 15 in.; bore, 9 in.; length of hub, 18 in. Owing to the high cost of a new gear, a decision was made to refit the gear on the shaft. This necessitated turning the shaft and boring the gear, besides fitting a steel bushing with ⅜-in. walls, truing keyways in shaft and gear, fitting a new key, turning each end of hub for about 1⅜ in. and shrinking a steel band on each end of the hub, as indicated in the accompanying sketch.

The gear was cast in one piece and split at the hub, so that it could be clamped on the shaft. Wedges were driven at each end of the hub to spread it, but not enough to distort the rim of the wheel. Clamping bolts were drawn up tight, and a portable boring bar was employed, as no machine large enough to swing the gear was available. The bar was set up near the rim and face of the wheel, and checked for alignment after each cut, as the cutter on striking the keyway had a tendency to shift the bar. Ends of the hub were turned with the same set-up.

When the gear was put on the shaft, four clamping bolts, 1⅝ in. diameter, were drawn up tight, the bands shrunk on each end of the hub, and the clamping bolts then removed. Following these repairs the gear gave satisfactory service.



Detail of gear repair operation

Cost of Trolley-Locomotive Transportation

Morris J. Elsing

TROLLEY and storage-battery locomotive transportation are the commonest types of mechanical haulage in metal mines. Trolley-locomotive transportation is the older and more commonly used, but storage-battery locomotive transportation is becoming more efficient, and apparently has a field where its use is more desirable.

The following summaries of statistical data of the economic aspects of trolley-locomotive transportation are presented in the order of the magnitude of operations, commencing with the largest:

Alaska Juneau, Alaska, A.I.M.E., Vol. 72; I.C. 6168—From the grizzlies the ore falls into chute raises of 200 tons' capacity. Ore trains consist of 44, 10-ton, flat-bottomed box-type cars drawn by 18-ton, articulated, Baldwin-Westinghouse trolley locomotives. Track gage is 30 in. and 50-lb. rails are used. Chute openings, 6 ft. long and 3 ft. high, are equipped with underslung, air-operated gates. Tramming distance to the mill is 2 miles. Cost of power in 1928 was 0.44c. per kilowatt-hour. Total cost of transportation in 1928, when 3,670,910 tons of ore was trammed, was 11.4c. per ton, or at the rate of 5.7c. per ton-mile.

Alaska Gastineau, Alaska, A.I.M.E., Vol. 63—Storage-battery locomotives gathered the ore on intermediate levels and trammed it to ore-way chutes at a cost of 6.7c. per ton. From these ore ways on the Sheep Creek tunnel it was trammed 3.5 miles to the coarse-crushing plant, 2 miles of this distance being underground. Eighteen-ton, Baldwin-Westinghouse trolley locomotives hauled 12-ton steel gondola cars over 50-lb. rails set with a gage of 36 in. Chutes were equipped with underswung arc gates, 6 ft. 8 in. wide, and also a check gate. The loading crew, consisting of a motorman and three chutemen who loaded the train in from 10 to 15 min., ran the train one-half mile to siding No. 2, where it was turned over to the running crew, which consisted of a motorman and two brakemen. Fifteen minutes was required to run the train 3 miles to the mill, 20 min. to dump in a 4-car tipple, and 20 min. to return to siding No. 2, where the next train was waiting. The two crews of seven men loaded, hauled 3.5 miles, and dumped 3,800 to 4,300 tons of ore in 8 hours. Costs per ton, 1915-18, were: Labor, $0.012; power, $0.001; maintenance of equipment and track, $0.017; general expenses, $0.006; total, $0.036. This was at the rate of 1.0c. per ton-mile. Tramming on the upper levels cost 6.7c., so the total cost of transportation was 10.5c. per ton, or at the rate of about 3c. per ton-mile. These costs apply to the transportation of 6,523,873 tons of ore, hauled during 1915 to 1918 inclusive.

Clifton-Morenci District, Arizona, A.I.M.E., Vol. 70—This haulage system was designed to handle 4,000 tons of ore per day, collected from the various mines of the company. The longest haul, from the Coronado mine, a distance of 4.2 miles, included a distance of 5,000 ft. underground at the Coronado mine, then 6,600 ft. on the surface, and then underground to the No. 6 shaft, where the ore was hoisted in skips and dumped into the concentrator bins. Trains consisted of 30 10-ton, gable-bottomed, side-dump cars, equipped with Westinghouse air brakes. Trains were drawn by three Jeffrey trolley locomotives, triple-connected with one set of controls. The gage of the track was 20 in.; 6x8-in. ties, 4 ft. long, spaced at 30-in. centers, were used with 50-lb. rails. The tunnels were 8x9 ft. in cross-section.

With a haul of 1.07 miles, studies showed the following time distribution of operations: Loading, 29 per cent; hauling, 24 per cent; unloading, 14 per cent; weighing and idle, 7 per cent each; switching, 9 per cent; and miscellaneous, 10 per cent.

Time studies gave the following averages for the Humboldt mine: To load 30 10-ton cars at mine chutes, 35 min.; to weigh 30 10-ton cars at scales, 7 min.; to dump 30 10-ton cars at pockets, 19 min. Running speed of the train varied from 5 to 7.5 miles per hour.

Costs for 1921 are given in Table I. These include loading, weighing, dumping, and all expenses except repairs to tunnels and loading chutes.

The following formulas were developed to apportion the total cost of train operation for varying lengths of haul from the principal mines where C is a constant equal to the operating cost when the distance is one mile:

Cost per wet ton hauled $= C$ [0.77 plus (0.23 x distance)].

Cost per wet-ton-mile $= C$ [0.77 plus (0.23 x distance)].

United Verde, Jerome, Ariz., A.I.M.E., Vol. 66, Hopewell Tunnel 1918—Ore from the shaft storage bins was trammed on the Hopewell Tunnel level (1,000-ft. level) to the Hopewell crushing plant, a distance of 8,900 ft. Trains consisted of fourteen standard-gage bottom-dump cars of 220-cu.ft. nominal capacity, but they were usually loaded to 280 cu.ft., or 20 tons of massive sulphide ore. Twenty 5-ton Baldwin-Westinghouse trolley locomotives with automatic air brakes were used. Drawbar pull was 12,500 lb. at 7.1 miles per hour. Cost of repairs was high because it included charges for equipping cars with airbrakes.

Costs are as follows: Labor, $0.054; supplies, $0.001; power, $0.005; repairs, $0.035; total, $0.095; credit limestone haulage, $0.005; net cost per ton, $0.090; cost per ton-mile, $0.053.

Costs per ton in 1926, according to *Mining Congress Journal*, September, 1927, were: Labor, $0.033; supplies, $0.001; electric power, $0.008; compressed air, $0.005; repairs, $0.025; total, $0.072; cost per ton-mile, $0.036.

Miami Copper, Miami, Ariz., A.I.M.E., T.P. 314—Ore is trammed from the transfer raises to shaft pockets in trains of 35 cars of 86-cu.ft. capacity, or 3.6 tons of ore. Trains are pulled by two 6-ton trolley locomotives in tandem. Train crew consists of one motorman and two brakemen. Drawbar pull of the locomotive is 3,000 lb., and the rated speed is 6.5 miles per hour. Cars are of the gable-bottomed side-

Table I—Costs for 1921, Clifton-Morenci District

	Humboldt	Clay	Coronado
Average daily tonnage	2,076	302	301
Average haul in miles	0.63	1.10	4.21
Cost per wet-ton-mile:			
Operating train	$0.018	$0.012	$0.005
General incl. supervision	0.004	0.003	.....
Cleaning tracks and blasting chutes	0.005	0.006	0.004
Track maintenance	0.007	0.007	0.009
Trolley line maintenance	0.002	0.002	0.002
Locomotive maintenance	0.003	0.003	0.002
Car maintenance	0.005	0.005	0.004
Block signal maintenance	0.001	0.001	0.001
Power	0.007	0.007	0.007
Totals	$0.052	$0.046	$0.034

This is one of a series of articles on the cost of metal-mine transportation in the United States.

dump type; when unloaded, they are unlocked by one brakeman and locked by the other while passing over the pockets at the rate of about 2 miles per hour. Track gage is 24 in. and rails weigh 45 lb. Loading chutes are equipped with steel, arc-type gates. Train movements, under the supervision of a train dispatcher, are controlled by 40 hand-operated signal lights. The average haul is 0.96 miles one way. To load the train requires 30 min. and 26 min. is required to haul, dump, and return, or a total of 56 min. Each train hauls between 950 and 1,000 tons per shift. Total production per day is 18,000 tons, which requires the use of six trains on each of the three shifts. One service train on each shift is required for hauling development muck and supplies. Cost of ore haulage, including loading, hauling, dumping, maintenance of track, equipment, and haulage drifts, is about 5c. per ton-mile.

Inspiration Copper, Miami, Ariz., M.C.J., Feb. 1, 1928—Compressed air and trolley locomotives are used. In the Live Oak section, 8-ton, 80-hp., 250-volt, direct-current trolley locomotives haul trains of 5-ton box-type cars, which are unloaded by a five-car dumping tipple. The train crew consists of a motorman and two helpers. Costs (Table II) do not include operating labor.

Climax Molybdenum, Colorado, E.&M.J., November, 1932—Trolley locomotives haul trains of ten or more 9-ton Card cars, equipped with Timken roller bearings. Length of haul is 6,300 ft., and sixteen trips are made daily. Cost of transportation, including power, maintenance, repairs, lubrication, loading, dumping, and every item chargeable to haulage, is 1.5c. per ton-mile.

Utah Copper, Bingham Canyon, Utah, A.I.M.E., Vol. 54—In 1912, when a large production came from underground workings, the main-level track was laid with 60-lb. rails, with 36-in. gage and 0.25 per cent grade in favor of the loaded cars. Two G.E. 10-ton locomotives, each pulling a train of seven cars, or 60 tons of ore per trip, handled as much as 4,000 tons per shift of 8 hours, the average being 2,500 tons, or 1,250 tons per train. If loaded from a chute in which the ore ran freely, a train could make a round trip in 15 min., divided as follows: Loading train at ore chute, 2.5 min.; trip to ore bin (4,000 ft.), 5.0; dumping train at ore bin, 2.5; return trip to chute, 5.0; total, 15.0 min.

Motor haulage in 1912 cost 2.5c. per ton, or at the rate of 3.3c. per ton-mile.

Fresnillo, Zacatecas, Mexico, A.I.M.E., Vol. 72—Ore is trammed through an adit an average distance of 2,000 ft. to the mill. A train of nine cars is drawn by an 8-ton, Baldwin-Westinghouse trolley locomotive. Cars have a rigid, flat-bottom body and hold 275 cu.ft. ore. They have swivel couplings and are unloaded in a one-car rotary car-dumper without uncoupling from the train. Track is 36-in. gage laid with 50-lb. rails, on 6 by 8-in. by 6-ft. ties, spaced 24 in. apart on a rock-ballasted roadbed with a 0.4 per cent grade in favor of the load. Motors have a rated drawbar pull of 2,000 lb. at a speed of 6 miles per hour. Two of these motors have hauled 1,600,000 tons and have traveled a distance of 13,600 miles with only nominal repairs. Cars are equipped with 4x7-in. Hyatt roller bearings; drawbars with 1-in. steel springs. The truck has four wheels, 20 in. diameter, made of chilled cast iron and pressed on 4.5-in. open-hearth steel axles. The train crew consists of a motorman and two brakemen. One chute tender assists in loading the two trains in operation. In addition to the train crew, one car dumper, one switchman, three repair timbermen, one track cleaner, and a shift boss (part time) are employed in transportation. The production was 3,500 tons per day and the cost per ton-mile was 8.2c.

Later information relative to Fresnillo, in I. C. 6661, states that after ten years of service, during which time three locomotives and 22 cars have handled 11,100,000 tons of ore, cars and locomotives are as good as new from the standpoint of the service they give. Such service is the result of adequate strength and a high standard of maintenance. Cost of transportation from July 1, 1921, to June 30, 1925, when 3,858,688 tons of ore was trammed, was 4.3c. per ton; this includes labor, 2.5c. per ton; supervision, 0.1c.; power, 0.6c.; and supplies, 1.1c. per ton.

Data in Table IV, relative to trolley transportation in 1927, was furnished me by the Westinghouse Electric & Manufacturing Company. In each instance trains were drawn by an 18-ton locomotive. A summary of cost data is tabulated in Table V. The mines mentioned represent the large operating units. Trains and cars are of large size, and the distance the ore is transported varies from about one-half mile to more than 4 miles.

The following review of essential transportation data is given for several of the smaller transportation systems.

Anaconda Copper, Butte, Mont., A.I.M.E., Vol. 68—Trolley locomotives, each weighing 3.5 tons, hauled ten loaded cars of 14-cu.ft. capacity, each car holding 0.9 ton of ore. Average haul was 1,500 ft. Normally, expectation was that a motor crew would haul 100 tons of ore in an 8-hour shift. In addition, some service haulage was required. Average number of loaded cars hauled per shift was 122.5, or 110 tons of material.

Mount Hope Mine, Mount Hope, N. J., E.&M.J., Sept. 8, 1930—Ore is transported from the chutes to the shaft pockets in trains of seven 5-ton, side-dump, Granby-type cars, drawn by 250-volt, direct-current, trolley locomotives. Crew consists of a motorman and one

Table II—Inspiration Trolley Locomotive Costs, Cents per Ton-Mile

(Operating labor not included)

	1926	1927	Total
Locomotive supplies	0.042	0.026	0.034
Locomotive repairs	0.304	0.210	0.257
Maintenance, trolley and rail bonds	0.059	0.156	0.108
Maintenance, motor generators	0.049	0.037	0.043
Power	0.546	0.329	0.437
Totals	1.000	0.758	0.879
Kilowatt-hour per ton-mile	0.661	0.442	0.476
Tons hauled (000 omitted)	1,224	3,953	2,588
Average distance in miles	0.332	0.553	0.500

Table III—Time Study of Tramming at Fresnillo

	Minutes
Loading	11
Transporting to switch	6
Waiting at switch	3
Switch to mill, dumping and return	25
Returning empties to loading chutes	5
Total for round trip	50

Tramming Costs per Dry Ton Hauled

Operation	$0.017
Cars	0.007
Track	0.002
Motor and generator	0.001
Car dumper	0.001
Trolley lines	0.002
Loading chutes	0.004
Total	$0.034

Recapitulation	
Labor	$0.021
Supplies	0.008
Power	0.005
Total	$0.034

Table IV—Trolley System Transportation Costs in 1927

Mine No.	Daily Tonnage	Haul in Miles	Maintenance per Ton-Mile, Cents	Total Cost per Ton-Mile, Cents
1	—	3.5	0.1	1.0
2	—	3.5	0.1	0.8
3	2,000	0.6	0.3	5.2
4	300	1.1	0.3	4.7
5	300	4.2	0.3	3.4
Average		2.6	0.2	3.0

Table V—Cost Data of the Larger Trolley Transportation Systems

Mine	Size of Cars, Tons	Tons per Train	Length of Haul, Feet	Cost per Ton, Cents	Cost per Ton-Mile, Cents	Year
Alaska Juneau	10.0	440	10,560	11.4	5.7	1928
Alaska Gastineau	12.0	400	18,480	3.6	1.0	1915–18
Clifton-Morenci	10.0	300	22,176	14.3	3.4	1921
Clifton-Morenci	10.0	300	5,808	5.1	4.6	1921
Clifton-Morenci	10.0	300	3,326	3.3	5.2	1921
United Verde	20.0	280	8,900	9.0	5.3	1918
United Verde	20.0	280	8,900	7.2	3.6	1926
Miami Copper	3.6	126	5,069	5.0	5.0	1928
Climax Molybdenum	9.0	90	6,300	1.3	1.5	1932
Utah Copper	9.0	60	4,000	2.5	3.3	1912
Fresnillo	14.0	126	2,200	3.4	8.2	1922
Westinghouse data	—	—	13,700	—	3.0	1927

helper. Average haul is 2,650 ft. and the maximum is 3,750 ft. Average time per round trip is 35 min. Cost of car loading is 2.1c. per short ton. The ore is magnetite, which runs 13 cu.ft. per ton, broken. Production in 1929 was 212,340 long tons. Total cost of delivering ore from chutes to the shaft pockets, including maintenance and car loading, was 6.4c. per short ton, or 12.8c. per short-ton-mile.

Old Bed & Harmony, Mineville, N. Y., I.C. 6092—On one of the main haulageways between auxiliary shaft pockets and main shaft 3,000-lb. drawbar-pull, trolley locomotives are used. Trains are made up of 3.5-ton cars. Average haul was 1,032 ft., and the amount hauled in one month in 1927 was 25,988 long tons, at a cost of 8.4c. per long-ton-mile, or 7.6c. per short-ton-mile. Haulage in 1927, as distinguished from tramming, cost 5.9c. per short ton, of which labor cost 2.5c.; supervision, 0.2c.; compressed air and drill steel, 1.4c.; electric power, 0.7c.; and other supplies, 1.1c. per ton.

Smuggler Union, Telluride, Colo., E.&M.J., Oct. 23, 1930—The Pennsylvania tunnel contained many crooked, narrow parts and steep grades. As a consequence of these conditions and because chute blasting was charged to tramming, costs per ton for tramming were high. The following figures refer to the period from March to August, 1927: Loading and motorman, contract labor, $0.119 per ton; chute blasting—labor, $0.026; ammunition, $0.010; track maintenance, $0.024; car maintenance, $0.022; motor maintenance, $0.026; power, $0.004; dumpers, $0.046; total, $0.277 per ton. Tons of unsorted ore totaled 93,628; tons waste, 1,716; distance trammed, 6,600 ft. Equipment consisted of two 4-ton 18-in. gage, 250-volt, direct-current trolley locomotives, and 30 3-ton, center-splitting, swivel-truck cars. Cost per ton-mile was 22.1c.

Cost of motor sublevel haulage from March to August, 1927, was as follows: contract labor, $0.190 per ton; chute blasting, $0.023; motor repairs, $0.027; car repairs, $0.023; power, $0.004; general overhead, $0.014; total, $0.281. Total tons amounted to 34,475; average distance trammed, 884 ft. Contract labor includes car dumping. Equipment consisted of 2.5-ton Jeffrey trolley locomotives and 3-ton, swivel-truck, center-splitting cars.

Copper Queen, Bisbee, Ariz., A.I.M.E. Vol. 52—Data are given in Table VI. Average distance trammed in 1914 was 2,939 ft., returning empty. Track and trolley extension costs were not included. In 1914 these amounted to 5.3c. per ton; transfer-chute construction cost 1.2c. per ton. The locomotives varied in size from 3 to 7 tons.

Copper Queen, Bisbee, Ariz., A.I.M.E., Vol. 48—In 1912, transportation amounted to 408,000 ton-miles. Maintenance costs per ton-mile during this period were as follows: Locomotives, 2.95c.; car, 1.64c.; track, 5.24c.; trolley, 3.60c.; power, 1.64c.

Magma Copper, Superior, Ariz., I.C. 6168—Ore is hauled on the 500-ft. level adit tunnel from the No. 3 shaft to the mill bins by 4-ton trolley locomotives of 2,000-lb. drawbar pull at a speed of 5 miles per hour. Cars, gable-bottomed, have a capacity of 3.6 tons. Distance to the mill is 4,000 ft., of which 1,700 ft. is underground. Direct current at 135 volts is used. Air- and hand-operated gates are used at the loading pockets. The crew consists of a motorman and two helpers. From 800 to 1,000 tons is handled per shift.

On the deeper levels the ore is transported from the chutes to the shaft in trains of ten 2-ton, gable-bottomed cars drawn by trolley locomotives. The crew consists of a motorman, chute tapper, and brakeman. Average haul is 1,000 ft.; from 300 to 400 tons is handled per 8-hour shift.

Mine No. 8, Southeast Missouri, I.C. 6160—Four-ton, trolley-haulage locomotives, powered by 250-volt direct-current double-unit motors, direct-geared to the drivers, are used for gathering and transporting ore to the shaft-skip pocket. These motors travel over 30-lb. rails with 24-in. gage. Track grades are kept below 4.0 per cent, and the motor is thus able to handle 8 to 14 loaded cars per trip. The motor delivers the empty 1-ton cars to a bypass or dead-end track, usually within a few yards of the muck pile. Hand shovelers tram the cars from the empty track to the required position for loading, and then back to the full track when loaded. This is the only hand tramming in the mine. The 2-ton cars are spotted by the motor for loading, and again removed when loaded. Ore is dumped into the skip pocket at the shaft by means of a rotary dump or tipple, which handles three 1-ton cars or one 2-ton car at a time. Average tramming distance is about 600 ft. Production for 1928 was 168,089 tons. Underground haulage cost per ton in 1928 was as follows: Operating labor, $0.107; repair labor, $0.023; repair material, $0.013; supplies, $0.020; power, $0.014; total, $0.177.

United Verde, Jerome, Ariz., A.I.M.E., Vol. 66—Six-ton, Jeffrey, trolley-type locomotives are standard for all main-haulage levels. Trains usually consist of twenty 18-cu.ft. cars. Direct current of 250 volts is used. About 65,000 tons was trammed an average distance of 850 ft. Costs per ton for the period October-November, 1919, are as follows: Labor—motormen, $0.036; switchmen, $0.033; grizzly man at skip pockets, $0.015; total labor, $0.084; power, $0.006; depreciation, motor equipment, $0.006; repairs to motor and supplies, $0.003; maintenance cars and tracks, estimated, $0.008; total cost per ton, $0.107; total cost per ton-mile, $0.670.

A round trip required 49 min., of which 63 per cent was for loading, 12 per cent for hauling, 12 per cent for dumping, and 13 per cent for switching.

Lucky Tiger, El Tigre, Son., Mexico, A.I.M.E., Vol. 72—Trains consist of 10 to 30 32-cu.ft., gable-bottomed cars, or 16-cu.ft. rotary dump cars drawn by 3-ton trolley locomotives. Distance from the shaft pockets to the bins is 6,000 ft.; production, about 200 tons per day. Tramming is done on contract at 5c. per ton for loading, tramming, and dumping. Total cost, including the contract labor, power, and repairs, in 1924, was 13c. per ton, or at the rate of 12c. per ton-mile.

Table VI—Ore Haulage by Trolley Locomotives at Copper Queen

	Cents per Ton	
	1914	Four Months 1915
Train labor	6.0	3.5
Loading labor	4.0	4.5
Motor maintenance	2.0	1.1
Car maintenance	2.2	1.3
Trolley maintenance	1.2	0.6
Track maintenance	2.0	1.4
Transfer chutes	2.2	2.8
Power	1.3	1.3
Miscellaneous	1.1	2.2
Foremen and bosses	1.0	0.9
Total per ton	23.0	19.6
Total per ton-mile	41.4	35.2

Pilares, Nacozari, Son., Mexico, I.C. 6307—Trolley and storage-battery locomotives handle waste and ore in different levels. Cars, 22- and 33-cu.ft. capacity, rocker-dump type, rated at a capacity of 1 ton and 1.5 tons, are used on levels below the 700; 16-lb. rails are laid on all levels except the 700-ft. main haulage level. The motor crew consists of two men on contract, who load at the chutes, haul, and dump the ore at the main ore dumps at the Guadalupe shaft. Costs of transportation per ton-mile by both types of locomotives are as follows: Trolley—1929, $1.11; 1928, $0.95; 1927, $0.92; storage battery costs—1929, $0.63; 1928, $0.71; 1927, $0.99.

Table VIII—Cost Data of Small Trolley-Transportation Systems

Mine	Size of Cars, Tons	Tons per Train	Tons per Shift	Length of Haul, Feet	Cost per Ton, Cents	Year
Anaconda	0.9	90	110	1,500	9.0 (a)	1921
Mount Hope	5.0	35	400	2,650	6.4	1929
Old Bed	3.5	..	...	1,032	1.5 (b)	1927
Smuggler Union	3.0	..	...	6,600	27.7	1927
Smuggler Union	3.0	..	...	884	28.1	1927
Copper Queen	...	..	..	2,939	19.6	1915
Magma	3.6	..	900	4,000	1.6 (a)	1928
Magma	2.0	20	350	1,000	4.1 (a)	1928
No. 8 S. E. Mo.	1.5	16	...	600	17.7	1928
United Verde	1.0	20	...	850	10.7	1919
Lucky Tiger	1.5	30	200	6,000	13.0 (c)	1924
Colorada	7.0	84	1,000	3,200	0.9 (a)	1929
Colorada	4.0	40	240	2,000	2.7 (a)	1929
Colorada	4.0	32	175	4,200	3.7 (a)	1929

(a) Estimated labor cost only. (b) This cost is for one particular haulageway. The average transportation cost for the mine was 5.9c. per short ton. (c) Contract labor cost is 5c. per ton.

Colorada, Cananea, Son., Mexico, I.C. 6247—On the 500-ft. tunnel level, ore is trammed from the shaft ore pockets to the railroad ore bins, a distance of 3,200 ft., of which 775 ft. is on the surface. Ore cars, bottom-dump type, have a level capacity of 135 cu.ft. and hold 7 tons of ore. Twelve cars form a train, which is drawn by a 10-ton, two-motored, center-control locomotive. The track is of 30-lb. rails laid to a 36-in. gage on treated 6 by 8 in. by 6 ft. ties, spaced 2 ft. center to center. The train crew consists of a motorman and a helper or two helpers. A man stationed at the chutes and another at the ore bins clean the drifts and grizzlies, assist in loading and unloading, and act as train dispatchers. Average output trammed is 1,000 tons per shift.

About 900 tons of waste is mined daily in glory holes at the Kirk mine and at the 5-10 adit. Waste cars are of the rocker-dump type, hold from 3.5 to 4 tons of waste, and are drawn by a 10-ton trolley locomotive. The waste

Table VII—Small Trolley Locomotive Costs

	Arizona	Montana
Weight, locomotive, tons	3.0	3.5
Capacity of cars, pounds	1,800	1,800
	—Cents per Ton—	
Power	2.3	2.2
Labor and material	19.8	35.0
Total operating	22.1	37.2
Locomotive maintenance		1.8
Car maintenance		7.5
Track, trolley, station maint.		11.0
Total		56.5

train crew consists of a motorman and two helpers; the Kirk waste train is made up of six to ten cars, which are loaded one at a time from ordinary stope chutes. About 50 cars containing 175 tons are trammed 4,200 ft. and dumped per 8-hour shift. Most of this distance is against an adverse grade. This crew also handles ten to twenty truck loads of timber and supplies.

The 5-10 adit waste train consists of ten cars, which are loaded two at a time from below bulldozing chambers. Length of the haul is 2,000 ft., and 60 cars containing 240 tons are trammed and dumped per shift. The crew consists of a motorman and two helpers. The 800-ft. level ore trains are in charge of a motorman, two helpers, and a man at the shaft who assists in unloading. Average haul is 800 ft.

Westinghouse Electric & Manufacturing Company has furnished me with costs at mines in Arizona and Montana where small trolley locomotives are used, which are given in Table VII. Data relative to such smaller unit transportation systems are tabulated in Table VIII.

Cost Analysis of Trolley Transportation—Most forms of transportation comprise three distinct operations—loading, tramming, and dumping. In trolley-locomotive transportation, the labor cost for each of these items varies greatly at the different mines. An approximate distribution of the time of the train crew is shown in Table IX for six mines. Factors affecting the time required for performing each operation are numerous. The most important are (1) character of the ore and the method of mining; (2) character of the ground in tramming drifts; (3) method of loading and unloading; (4) size of cars; (5) number of cars in trains; and (6) length of the haul. Cost of actual tramming alone is a function of the distance. Loading and unloading costs are independent of the distance and are a function of the tonnage.

Table IX—Time Distribution of Trolley Locomotive Transportation (Minutes)

Mine	Length of Tram, Feet	—Loading— %	Min.	—Tramming— %	Min.	—Dumping— %	Min.	Miscellaneous %	Min.	Total Min.
Alaska Gastineau	18,480	18.0	20.0	63.0	70.0	18.0	20	....	....	110
Clifton-Morenci	5,650	29.0	35.0	24.0	19.0	14.0	....	33.0	....	..
United Verde	850	63.0	31.0	12.0	6.0	12.0	6	13.0	6.0	49
Miami	5,069	54.0	30.0	32.0	18.0	14.0	8	....	....	56
Utah	4,000	16.6	2.5	66.7	10.0	16.6	2.5	....	....	15
Fresnillo	2,000	22.0	11.0	54.0	27.0	24.0	12	....	....	50

Labor cost of hauling can be more accurately predicted than either of the other two items. Locomotives, generally, have a speed of from 5 to 7.5 miles per hour, with an average of about 6 miles per hour. Knowing the distance and the tonnage per train, with an allowance for delays, an accurate estimate of labor costs per ton can be made. Capacity of trains is an exceedingly important factor in determining this cost. This is apparent when one remembers that in some mines a train crew handles ten tons per trip, in others up to 440 tons, as at the Alaska Juneau.

Loading costs per ton, as pointed out in the discussion of the "Cost of Hand Tramming," depend largely on the character of the ore being drawn. It may have characteristics that are disadvantageous for efficient drawing, or it may be made so by the method of mining. This is usually the case with shrinkage mining, where bulldozing is always a large part of the cost. Bulldozing is not usually made a part of the transportation cost, but was so included at the Smuggler-Union, worked by the shrinkage method. Blasting of boulders in car loading is an important part of the cost reported for this mine.

To draw definite conclusions relative to average costs is rather difficult, but, from the data presented, several facts are apparent. Low costs are obtainable only when train capacity is great. Total cost of transportation of the larger operations averages about 5c. per ton-mile. Because of the erratic costs shown, no attempt is made to give an average for the smaller operations. In general, operating labor amounts to about 50 per cent of total cost; maintenance of equipment and track, to about 40 per cent; and power, to 10 per cent.

Cost of labor for the larger operations is divided approximately on the basis of 50, 35, and 15 per cent to loading, tramming, and dumping, respectively, when the distance is about a mile. For a number of these same companies, the power cost averages about ½c. per ton-mile.

■

Drilling to determine extent of a diamond bearing structure at Moreno, Brazil

Screen Testing Ores by Hand

IN THE EARLY WORK of the Technical Committee on Milling Methods of the A.I.M.E. it was found necessary to undertake the development of a standard laboratory method for screen testing of ores. During the research stages the desirability of having this standard, when completed, approved as the American standard was brought out. In accordance with this idea, its initiation as a project of the American Standards Association (then the American Engineering Standards Committee) was approved by the latter body and the A.I.M.E. designated as sponsor. In 1929, the A.S.A., on recommendations of its advisory committee, the Mining Standardization Correlating Committee, authorized the A.I.M.E. to be sole sponsor for this project. The personnel of the Institute's Committee on Milling Methods was then increased to make it thoroughly representative of all diversified interests in the mining field. Two years ago the standard was circulated, and it has been given wide distribution. Early in 1932 the committee voted that the standard be recommended to the board of directors of the Institute for submission to the American Standards Association for approval as American recommended practice. This was done, and the American Standards Association approved the standard.

Sieves—The sieves shall be 8 in. in diameter, with well-fitting pan and cover, all to be free from crevices wherein particles can lodge. The sieve-scale ratio must not be more than $\sqrt{2}$, the finest sieve used being optional with the individual operator. It is recommended, however, that sieving be carried to 400 mesh, the 200-mesh sieve being followed by the 270 (aperture 0.052 mm.) and this by the 400 (aperture 0.037 mm.).

Preparation of the Sample—The material to be sized should be thoroughly dried at 110 deg. C., well mixed, and divided by riffling to an amount within 10 per cent of the standard charge. The final adjustment of the exact weight may be made by adding to or subtracting from this split, although it is preferable to take the entire split.

Size of Sample or Weight of Charge—If the coarsest particles in the sample fall within the limits of size designated in the accompanying table, the corresponding weight of charge is considered as standard.

Double-size charges may be taken by making two sizing tests and combining like products.

Wet Splits—The sample is sieved wet through the finest sieve, preferably using distilled water. This may be done either by decanting through the sieve or by direct wet sieving on the sieve. This operation is carried only far enough to remove all of the slime and some sand. The undersize and oversize are both dried at 110 deg. C., cooled, and weighed. Any loss in weight is added to the undersize.

Table I—Standard Charge Weights

Limit of Size of Coarsest Particles in Sample, Mm.	Standard Charge, Grams
16.00—11.32	40,000
11.32— 8.00	12,500
8.00— 5.66	5,000
5.66— 4.00	2,000
4.00— 2.00	1,000
2.00— 1.00	500
1.00— 0.50	250
0.50— 0.25	100
0.25— 0.00	50

Standard Hand-sieving Manipulation—In dry hand-sieving, it is advisable to work over a smooth paper to indicate spills. The sieve with pan and cover in place is held slightly inclined in one hand, and with a stroke of 6 to 8 in., is gently struck against the other hand 150 times per minute, turning the sieve one-sixth revolution after each 25 strokes, thus completing a revolution in one minute. This operation is called a one-minute period in the standard. If the 400-mesh sieve is used, a 6-in. diameter is recommended; with a 6-in. sieve, 200 strokes, at the same speed, are given, turning the sieve one-eighth revolution after each 25 strokes. This cycle is the equivalent of 150 strokes for the 8-in. sieve.

End Point in Sieving—The end point in sieving is reached when less than 0.05 per cent of the weight of the original charge passes through the sieve in a one-minute period.

Dry Sieving Oversize From Wet Split—The oversize from the wet split, after drying and weighing, is sieved dry on the finest sieve until the undersize for a one-minute period is less than 0.1 per cent of the original weight of the charge. Brush both sides of the sieve to remove dust and loosely held particles in the meshes. Return the oversize to the sieve, and continue sieving until the undersize for a one-minute period is less than 0.05 per cent. Repeat the brushing of the sieve, and sieve for another one-minute period. If the undersize from this is again less than 0.05 per cent, return it to the oversize; if it is more than 0.05 per cent, continue sieving until a one-minute period gives less than 0.05 per cent. All of the undersize of the finest sieve is weighed and added to the undersize from the wet split.

Remove the oversize, and clean the sieve by tapping sides, brushing, and by rubbing with a flat rubber. The material obtained from the sieve by this cleaning is added to the oversize, which is weighed. Any loss in weight is added to the undersize of the particular sieving in which the loss occurred. The total loss for the entire sieving operation should not be more than 1 per cent of the weight of the original sample.

Optional methods, which may be used, but are not considered as standard, follow:

Separating the Finer-Than-Sieve Size—No method or apparatus is recommended as standard. Whether sedimentation or elutriation is employed, the operation must be carried on at a definite temperature, 20 deg. being suggested as a standard, and the sample must be thoroughly dispersed. Distilled water should preferably be used.

The following settling rates, based on Richards' figures for quartz (Richards' "Ore Dressing," Vol. IV. Appendix) are tentatively suggested as a standard to continue the $\sqrt{2}$ sieve scale series.

Sieving the Coarser Sizes First—After the wet split has been made on the finest sieve, the dried oversize may be sieved by starting with the coarsest sieve, the sieving to be carried to the end point as described under "Dry Sieving Oversize From Wet Split." In this

Table II—Tentative Settling Rates for Finer-Than-Sieve Size

Size Range Desired, Microns	Average Size, Microns	Falling Velocity Mm. per Sec. for Coarse	Fine	Time Required to Produce This Product for 1 Meter Fall
74(a) to 52	63.0	3.6	1.8	Settled Product for First 9 Min.
52.0 to 37(b)	44.5	1.8	1.0	9 to 17
37.0 to 26.0	31.5	1.0	0.5	17 to 33
26.0 to 18.5	22.2	0.5	0.22	33 to 76
18.5 to 13.0	15.8	0.22	0.13	76 to 128
13.0 to 9.2	11.1	0.13	0.08	128 to 210
9.2 to 6.5	7.9	0.08	0.05	210 to 330
6.5 to 4.6	5.6	0.05	0.03	330 to 560

(a) Equivalent to 200/270 mesh. (b) Equivalent to 270/400 mesh. The particles or each settling product may be measured by means of the microscope, from which the average size may be calculated. No method for the microscopic measurement is recommended as a standard. Finer-than-sieve-size products, if not actually measured, are reported as products settled in one meter for a definite time period.

case the loss in weight must be reported as "loss in weight," but is included in the percentage of undersize from the finest sieve.

Dry Sieving—Material that undergoes disintegration or alteration when wetted must be sieved dry. In this operation the sieving is the same as described under "Dry Sieving Oversize From Wet Split." Each size, before being considered as completely sieved, must be lightly brushed on the sieve to free the particles from adhering dust.

As the separation of the finer-than-sieve size cannot be made in this case by settling in water, the measurement of this material may be made without separation as described by Perrott and Kinney (*Jnl. Ceramic Society*, Vol. 6, No. 2, February, 1923, pp. 417-439), in which the dry material is embedded between two slides in Canada balsam containing 20 per cent xylol, and a microscopic count and measurement made.

Wet Sieving—Material that undergoes change in size due to decrepitation upon drying at 110 deg. C. must be wet sized. It is not possible to start with a known weight of charge if a moisture determination is not feasible. In this case the final weights of the products must be considered as the original weight of charge, and the loss in sizing cannot be determined. As it is tedious to determine when the sieving has arrived at an end point, this must be determined by judgment of the amount passing through the sieve in one minute of sieving. As this is not sufficiently accurate for a standard, wet sizing is not recommended except where dry sieving is impracticable.

Machine sieving may be employed by sieving for one-minute intervals in lieu of the one-minute period in hand sieving. A sieving analysis may be made by machine, using a nest of sieves. The sample must first be wet split on the finest sieve. The sieving is carried on in three periods of 30 min., 20 min., and 10 min., respectively. After each period the sieves are cleaned and the progress of the sieving is checked by hand sieving as follows:

Starting with the coarsest size, the oversize, after weighing, is hand sieved for a one-minute period. If the undersize is less than 0.05 per cent of the original weight, the sieving for this size is considered complete; if the undersize is between 0.05 and 0.1 per cent, the sieving is completed by hand; if the undersize is more than 0.1 per cent, machine sieving is continued for 20 min. After the end of the 20-min. sieving period, the completion of the sieving is again checked by hand as before, and, if necessary, a third machine sieving of 10 min. is given. After the 10-min. sieving period, all sizes are finished by hand.

With this method the loss in weight must be reported as "loss in weight," but is included in the percentage of undersize from the finest sieve.

Standard Sizing Test Report—The data obtained in the sizing test are reported in a manner that will give the following information:

Place and date of test.

Sieves used.

Methods used: if standard, at what size the wet split was made; if optional, designate which optional method used.

Weight of charge used.

Nature of the material.

Whether the finer-than-sieve size was separated and, if so, indicate method employed.

Adjustment of sieving losses.

The tabulation of the results should include:

1. The limiting sieves for each sieved product, as 65/100, indicating minus 65, plus 100 mesh.

2. The sieve apertures expressed in microns (one thousandth millimeter).

3. The weight in grams for each product.

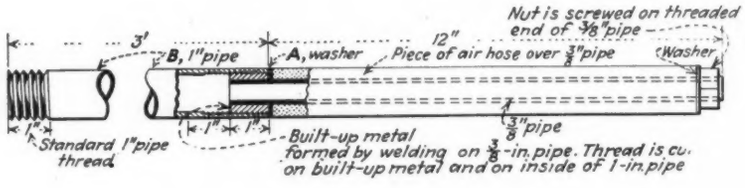
4. The percentage of each product based on the original weight of the charge.

Plug for Diamond-Drill Holes

DETAILS of a plug for stopping the flow of water in a diamond-drill hole made with a size E standard bit are shown in the accompanying sketch, contributed by Carl Trischka, Bisbee, Ariz.

Water under pressure and in considerable volume, writes Mr. Trischka, is often encountered in underground drilling operations. This water must either be held back or pumped to the surface. Makeshift devices, such as wooden plugs driven into the drill holes, are unsatisfactory. The plug shown herewith, which was invented by J. Valenzano, a diamond-drilling contractor of Bisbee, stops the flow of water immediately and permanently. Moreover, if the flow encountered makes further drilling impossible by usual means, a plug of a slightly larger size than that shown may be employed and drilling continued through the plug, which thus serves, also, as part of the casing.

Referring to the sketch, operation of the plug may be described as follows: The plug on being inserted into the hole fits snugly with the rock walls. Rotation of the 1-in. pipe *B* exerts a pressure on washer *A*, which pressure causes expansion of the air hose to form a tight fit with the rock walls of the hole. By closing a valve placed on the outer end of *B*, the flow of water is shut off, pressure of the water strengthening the bond between the hose and the walls.



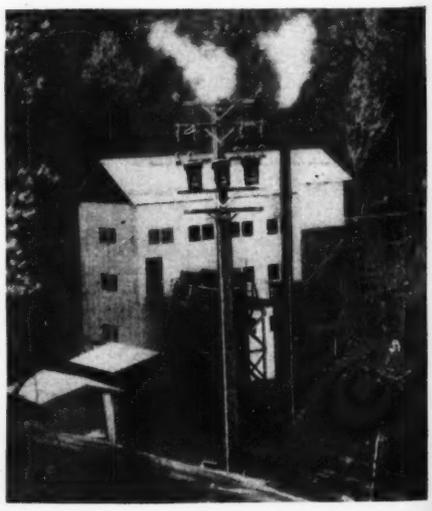
Quicksilver Production Costs

A TOTAL COST of $47.82 per flask is reported for 1932 by the Bumsted Mining Company, operating the Great Western quicksilver mine, near Middletown, Calif. Output during the year totaled 607 flasks, the plant treating 6,063 tons of ore in 315 working days for an average recovery of 7.6 lb. per ton. Detailed costs, on the basis of per ton of ore treated, follow:

Mining: Labor, $1.23; supplies, $0.17; total, $1.40.

Treatment: Labor, $0.74; supplies, $0.06; maintenance, $0.04; fuel oil, $0.49; power, $0.14; light, $0.03; flasks, $0.08; total, $1.59.

Fixed and General: Audit and legal, $0.06; insurance, general, $0.04; insurance, compensation, $0.22; interest and taxes, $0.19; assaying, $0.02; automobile and truck, $0.13; road repairs, $0.02; camp maintenance, $0.03; office and management, $0.62; depreciation, $0.30; depletion, $0.07; miscellaneous, $0.10; total, $1.80; grand total, $4.79.



Underground Scraping Equipment

Lucien Eaton

THE use of scrapers, often called drags or slushers, has become widespread during the past ten years. They are now used in mines, open-pit and underground, in most countries of the world. The simplicity, adaptability, capacity, and ease of operation of this type of loading equipment have made it popular, and in many districts it has entirely replaced other machines and methods of loading.

Scraping, or slushing, is similar to hoeing. The scraper takes the place of the blade of the hoe, and is moved forward and back by ropes attached to its front and rear, instead of by a handle. The scraper moves the dirt in front of it when going forward, and slides over it when going back. The hauling or digging rope is attached directly to one drum of a double-drum hoist, and the back haul or tail rope passes around a pulley, which is secured in the breast or behind the ground that is to be moved, and is attached to the other drum of the hoist. When cars are to be filled, the dirt is dragged up an inclined slide and is dropped through a hole into the car.

The original scraper used in mining work is shown in Fig. 65f and originated in the Tri-State zinc district near Joplin, Mo., where it is still used for rehandling tailing from the dumps. The next steps in development were a modification of the Bagley scraper and an adaptation of the Page dragline bucket, following which various designs and sizes of scrapers were evolved, until now three general types are recognized as standard equipment. Each has its own field, although the fields overlap to some extent.

For handling fine material, such as sand, gravel, crushed rock, and coal in pits or storage piles, a scraper of the Crescent or LeClair type, as sold by Sauerman Brothers, of Chicago, Ill., is usually best. The upper part of the blade is bent forward over the load, so that as soon as the scraper is full it ceases to dig, and "rides the pile." This type of scraper requires a movement of about 20 ft. to pick up its load, and is therefore not suitable for use underground where the breast must be cleaned out; but it may be used in fine ore in transfer drifts.

The second type is the box scraper, used underground in fine ore and easy digging, and in its large sizes on long hauls in coarse and heavy ore. This scraper has parallel sides, held apart at the front by a crossbeam. Across the back is a curved, slanting blade, which does the digging. Scrapers of this type are as small as 30 in. wide and as large as 60 in., and vary in weight from 150 to 3,000 lb. The standard scraper for sublevel work is 36 to 42 in. wide, has a capacity of 8 to 10 cu.ft., and weighs about 450 lb. Only the larger sizes, used for loading coarse ore, have teeth attached to the blade.

The third type, which bids fair to replace the box scraper in most underground work, is the hoe scraper. This has a curved blade, which may be straight across the back or crescent shaped, carried by a bale, attached to the top of the blade near the ends, and coming together at the front, where the rope is attached. Steel rails have been used a great deal as material for the bail, but they fail in severe service, and in the later designs the bails are made of special steel castings or standard bars. Many hoe scrapers have short side plates or "cheek plates" at the ends of the blade, which help to prevent spill at the ends without materially interfering with digging in coarse material. A few double-bladed hoe scrapers are available, but their use is limited.

Hoe scrapers are designed especially for handling coarse and heavy material, and for this reason are relatively heavy. They are standard for loading with slides in drifts and tunnels, and for most work in stopes and on sublevels. The standard size for drifts and sublevels is 42 in. wide, has a capacity of 8 cu.ft., and weighs 600 lb. For heavy service they may be as wide as 72 in. and weigh as much as 2,800 lb.

One of the most successful of the newer designs has a blade curved like a crescent, and has no cheek plates. This type, developed at the Mascot mine of the American Zinc Company at Mascot, Tenn., is especially effective in coarse ore.

The angle that the digging edge of the blade makes with the horizontal depends somewhat on the material to be moved, and is greater for coarse ore than for fine. Maximum digging effect is usually about 30 deg., but digging is not always the most important consideration, and in coarse ore the angle may sometimes be increased to 45 deg. with advantage. The upper part of the blade should be curved over toward the front, so that when the scraper picks up its load the pressure of the dirt on this part of the blade will prevent it from burying itself. If the blade is curved on a radius of 16 to 18 in., the desired effect is obtained.

Scraper blades are usually made of boiler plate, ½ to 1 in. thick, and the digging edge is coated with stellite, a hard alloy that can be applied with an oxyacetylene blowpipe or an electric-arc welder, or is protected by a manganese-steel lip, which is riveted on; or the entire blade can be made of manganese steel. If the blade is made reversible, so that full wear can be obtained from both edges, and if manganese steel can be obtained for less than 14c. a pound, solid manganese-steel blades are the most economical. Worn-out blades can be cut up and used as chute liners.

At first, small double-drum hoists of 5 to 10 hp., driven by compressed air, were largely used as scraper hoists, but these proved inadequate, and in standard equipment the minimum is now 15

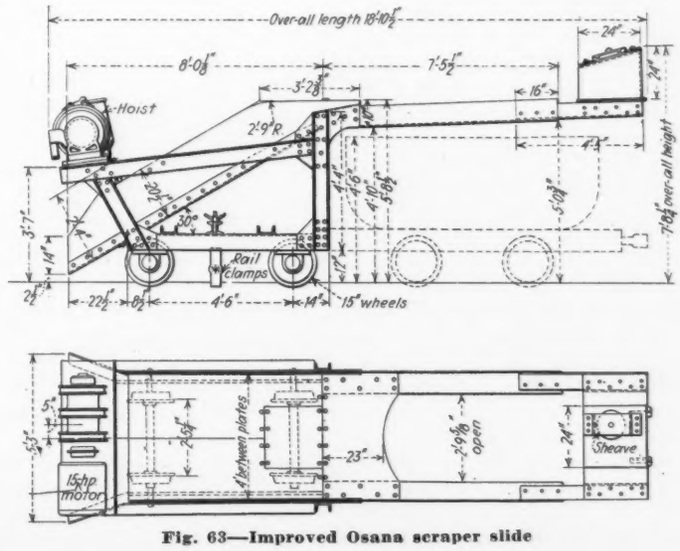


Fig. 63—Improved Osana scraper slide

One of a series of articles on mine equipment.

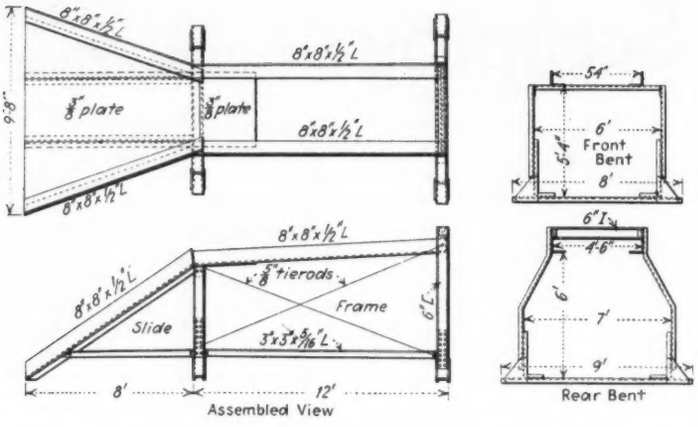


Fig. 64a—Semi-portable slide, Cliffs Shaft mine

hp. and from this they range up to 75 hp. Most hoists are now driven by electricity. The drums are ordinarily placed side by side, but tandem drums are in favor in some places. A few three-drum hoists are available, as in the Goodman patented system of loading, the third drum being used for shifting the position of the tailblock when loading around a corner.

For underground use the hoist must be simple, compact, and self-contained, and its size and weight must be kept within reasonable limits. The drums are therefore smaller than those used in regular hoisting practice. Drums of larger diameter reduce rope wear, but increase weight and require greater gear reduction; drums of small diameter increase the bending stresses in the rope, and narrow drums increase frictional wear on the rope. On a 15-hp. hoist the minimum drum diameter should be 8 in.; and the minimum width of face 6 in. Drums, bearings, and base plates or frames are made of cast iron, cast steel, or welded steel. The latter type of construction is gaining in favor, on account of its light weight and low cost.

Speed reduction is usually accomplished by spur gearing, and less often by worm gearing or by chain and sprocket. Hardened, tool-steel, cut gears only should be used. If worm gearing is used, it should run continuously in one direction. Chain-and-sprocket drive permits compact design, but is limited to moderate ratios of reduction, because small chains will not survive, and there should be not fewer than thirteen teeth in the smallest sprocket. Steel roller-bushed chain $\frac{3}{4}$x$\frac{3}{4}$x$1\frac{1}{2}$ in. gives satisfactory service on 25-hp. hoists, and should be not less than $\frac{5}{8}$x$\frac{5}{8}$x$1\frac{1}{4}$ in. for 15-hp. hoists. Chains should run continuously in one direction.

Clutches of various kinds have been tried. Automatic positive clutches were used with some of the small dirt hoists, but when these are used on electric hoists, the motor must be continually stopped and reversed, and they are not suitable for the larger sizes. Friction clutches of several types have been tried. Of these, a band clutch in connection with planetary transmission seems to be the most suitable for the service required. Both drums revolve in the same direction, and both can be operated at the same time, if desired.

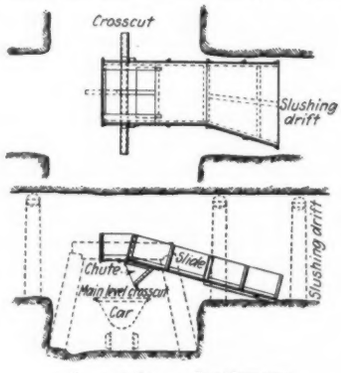


Fig. 64b — Stationary scraper slide, Montreal mine

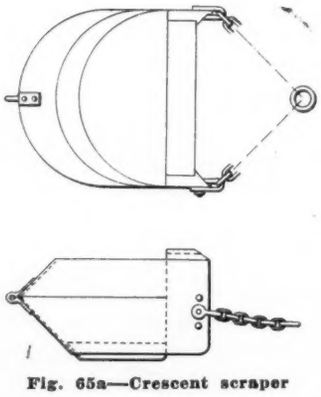


Fig. 65a—Crescent scraper

This is of great assistance in righting heavy scrapers that have been overturned, and in pulling scrapers out of chutes.

In some kinds of work, such as loading cars with a slide, to reduce the speed of the scraper is desirable at times, as when it is dragged up the slide and when it is picking up its load at the face. This is usually done by slipping the clutch; but if carried to excess it results in overheating and poor lubrication. To overcome this, a second spider, having a higher gear ratio, may be geared to the hauling drum, and be used when slow speed or greater pull is wanted. At slow rope speeds this is unnecessary. As to the question of rope speed a considerable difference of opinion exists. On short hauls under 75 ft. a speed of 150 ft. per minute is sufficient; and in picking up its load in coarse material the scraper must move even more slowly, so that it will have time to drop between the chunks. If the bottom, over which the scraper is dragged, is rough and uneven, the load will be spilled at high speeds; but if a smooth track is provided the speed may be increased to as much as 400 ft. per minute. A smooth track may be made by laying three or four lines of old rails or poles along the bottom of the drift. This is usually done in such places as transfer drifts, in which the broken ore from stopes is dragged to a central raise, instead of being loaded into cars and trammed. Higher speeds may also be used when scraping down grade than on a level floor, because spill is not so important. Ordinarily, the rope speed of both drums is the same, but sometimes the tail rope is run at a higher speed than the other. Whether or not this is desirable depends on local conditions, especially upon the slope up which the scraper must be dragged on the back haul. On the average, a large scraper moving slowly is more effective than a small one moving rapidly, power consumption being the same.

Scraper hoists and scraper outfits are made by a large number of firms, among which may be mentioned the Sullivan Machinery Company, Sauerman Brothers, and the Goodman Manufacturing Company, all of Chicago, Ill.; the Gardner-Denver Company, of Denver, Colo.; the Ingersoll-Rand Company, of New York; the Lake Shore Engine Works, of Marquette, Mich.; the St. Louis Power Shovel Company, of St. Louis, Mo.; Holman Brothers, of Camborne, England, and Fullerton Hodgart & Barclay, of Paisley, Scotland.

Excepting the labor cost of operation, the largest single item of expense

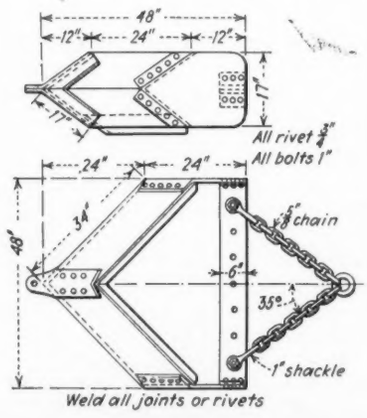


Fig. 65b—V scraper for fine material. Dimensions given are for ½-ton capacity

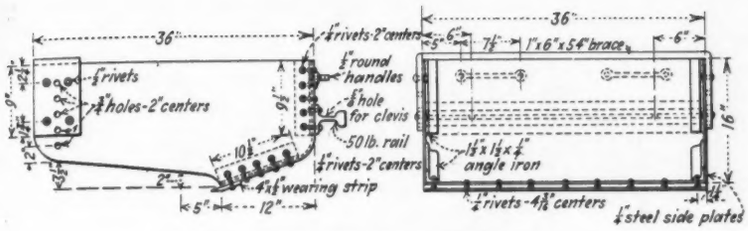


Fig. 65c—Box scraper

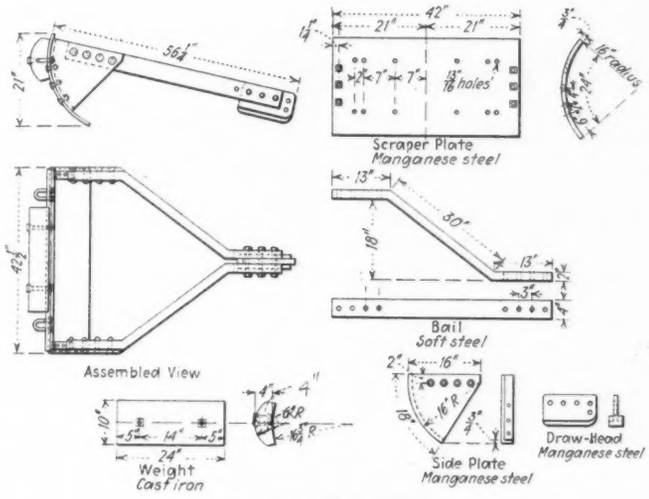


Fig. 65d—Hoe scraper

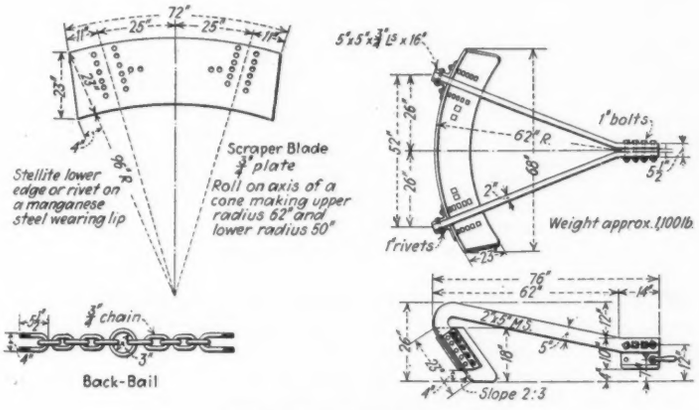
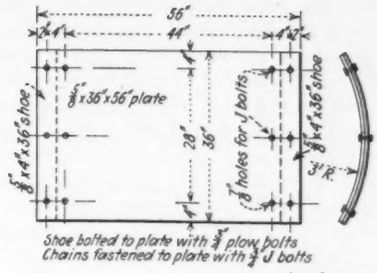


Fig. 65e—Mascot scraper



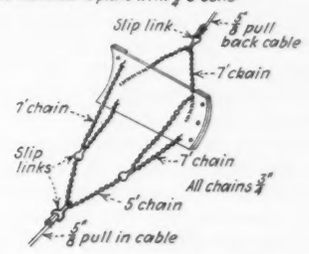


Fig. 65f—Original Tri-State scraper, capacity 24 cu.ft.

is usually that of rope replacement. Most of the wear is frictional, but some is due to bending stresses set up by the small sheaves used. Frictional wear occurs partly on the drum and partly on the floor of the drift or stope. Wear on the drum can be materially reduced by oiling the rope frequently and by keeping the hoist properly lined up with the path of the scraper. Wear in the stope can be reduced by care in preventing the rope from rubbing on chunks or on the walls, and all frictional wear is less on plow-steel rope than on the softer grades. Breakage of wires from bending is also less with the higher grades of rope. Soft ropes are often crushed and weakened when irregularly wound in multiple layers, and for this reason many operators prefer to have the rope laid up with an independent wire-rope center instead of a hemp center. The usual construction is regular lay with six strands of nineteen wires each.

If plow-steel rope is used, rope $\frac{3}{8}$ in. in diameter is suitable for 10-15-hp. hoists, $\frac{1}{2}$ in. for 20-hp., $\frac{5}{8}$ in. for 25-30-hp., $\frac{3}{4}$ in. for 35-50-hp., and $\frac{7}{8}$ in. for 60-75-hp. Some operators use a larger size for the hauling rope than for the back haul, but the economy of this is doubtful. The back haul or tail rope is twice the length of the hauling or digging rope, and lasts usually twice as long. When worn out, a part of it is generally still good, long enough to be used as a hauling rope if the two are of the same size and quality. The rope is usually attached to the scraper with a clevis and a thimble and Crosby clips; but a wedge socket, open or closed, occupies less space and is more quickly put on. To attach an open socket directly to the scraper is a mistake, however, because it does not afford enough flexibility. "Pigtails" and snap hooks have been used with small scrapers, but are not suitable for the larger equipment now in vogue.

The pulley block over which the tail rope passes is an important part of the scraping outfit. It must be reasonably light and very strong, and at the same time not too expensive. The service is much more severe than blocks of the same size are ordinarily expected to meet; and, consequently, most blocks not especially designed for it have been unsuccessful. Pressures developed in the bearings are high, and the maximum that the bearing should be expected to stand should be 1,000 lb. per square inch of projected bearing area. A pull of 2,000 lb. on the rope means a pressure of 4,000 lb. on the pulley, and requires a pin at least 2 in. diameter and 2 in. long. The pin should be fast in the frame, and the sheave should turn on it. Turned steel rod or cold rolled shafting makes a good pin. Standard brass pipe, cut off by a hacksaw and pressed into the sheave, makes cheap and satisfactory bushings.

The sheave should be made of cast alloy steel or manganese steel, and the groove should be turned or ground reasonably smooth. Cast-iron sheaves do not survive. The groove should fit the rope, but the sides should be flared at approximately 45 deg., because in use the sheave is seldom perfectly lined up, and the wide flare of the groove prevents excessive friction on the sides. Ordinarily, the sheave should be 2 in. wide. For convenience in handling, the block should be as light as the service will allow, and the diameter of the sheave in the groove should be at least twelve times the diameter of the rope. The frame must entirely cover the sides of the sheave, so that it will be impossible for a man to get more than the tips of his fingers caught be-

tween the rope and the sheave. The opening through which the rope passes should be large enough to pass a knot easily, and a guard should be provided to prevent the rope from crowding between the sheave and the frame. Side plates and frames are made of steel or malleable castings, or of boiler plate.

Small blocks for hoists not larger than 15 hp. may be hung from hooks, but the hooks should not be swiveled. The larger blocks should be hung from clevises, and should be so designed that they can be threaded with the rope without disconnecting it from the scraper.

A good many mining companies make their own blocks, and the designs shown in Fig. 67 are such as can be made in any mine machine shop. Special blocks for this service are sold by most of the companies selling scraper hoists.

In drifts and in many stopes in hard or reasonably firm ground the best device for securing the tail block is an eyebolt held in place by one or two outside or Lewis wedges. Split eyebolts having internal wedges should never be used when the pull is direct. They cannot be recovered, except by

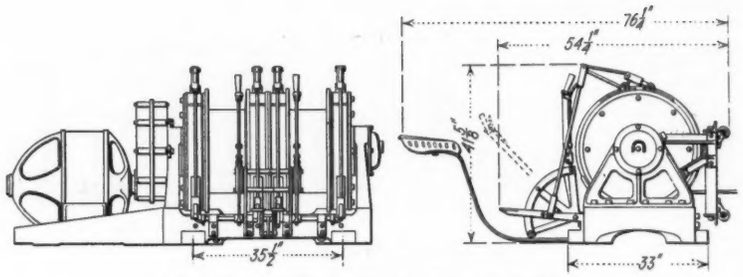


Fig. 66a—Sullivan 50-hp. scraper hoist

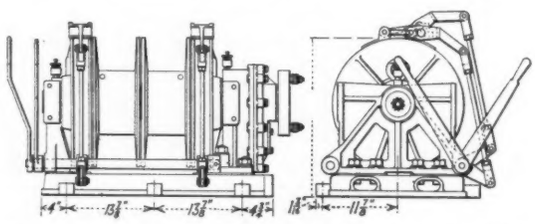


Fig. 66b — Holman 15-hp., double-drum scraper hoist

blasting, if they hold, and are dangerous if they come out. If the pull is not in line with the eyebolt, a flexible bolt should be used, such as is shown in Fig. 68c. This eyebolt, developed at the Mascot mine, at Mascot, Tenn., consists of a standard closed socket attached by a short piece of wire rope to another socket fitted with an outside wedge.

To use eyebolts in soft ground that requires timbering may be impossible, and in these cases the tail block is hung from a boom or sprag braced between the last cap and the breast. Booms of various kinds have been used, one of the most convenient designs being shown in Fig. 68d. A stoping bar should never be used for supporting the tail block, because if it bends it will be pulled out, and may cause a serious accident. In large stopes, the block may be hung on a wire rope or chain stretched between two eyebolts, but this is not usually as effective as flexible eyebolts.

Adequate lighting, essential for the safety and efficiency of the men, is secured in drifts usually with incandescent lamps on flexible two-conductor cable of the "cab-tyre" or Tyrex type, and in stopes with floodlights. Railroad type floodlights give good service if fitted with heat-resisting lenses; but good results are also obtained with home-made floodlights without lenses. A satisfactory light can be made by fitting a standard reflector and socket into one end of a short piece of sheet-iron tubing, or by fitting a socket in the bottom of a 14- or 16-qt. water pail.

Scraper slides, used for loading cars, may be classified into two main divisions: (1) movable slides, mounted on wheels; (2) stationary slides, attached more or less firmly to the ground. To the first division may be assigned the special slides attached directly to cars, as well as the more or less standardized

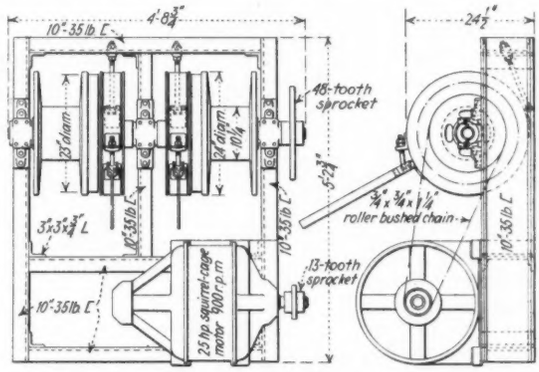


Fig. 66c—25-hp. chain-driven scraper hoist

mounted slides that carry the hoist but are independent of the cars. In the second division are included the slides built upon or attached directly to mine timbers, which are in reality chutes, the floors of which slope uphill instead of downhill. It also includes all chutes that must be taken apart or disconnected when they are moved. The first division is more numerous than the second.

Built at the Osana mine, near Iron River, Mich., the original successful movable slide is known as the Osana slide. This same design is also called the Eureka slide, because it was used later with great success at the Eureka mine, at Ramsay, Mich. The slide consists of a platform sloping toward the front, flared at the bottom, supported by a framework mounted on two sets of wheels. The framework also carries a double-drum hoist over the front part of the slide in such a position that the scraper can pass under it. Sides are attached to the stoping platform, and these are carried, horizontally and parallel, to the rear from the top of the incline, and support a horizontal plate in which is an opening through which the ore falls into a car. At the rear end a sheave is mounted, and the digging rope from the hoist passes over this sheave and back under the hoist to the scraper, which is pulled by it up the slide and over the hole in the floor. The tail rope passes around a sheave, hung in the breast of the drift, as is done in regular practice. This rope must pass out from the under side of the drum, unless the direction of rotation of the drums is reversed at each pull.

The slope of the inclined portion of the slide is 25 to 30 deg., and must be

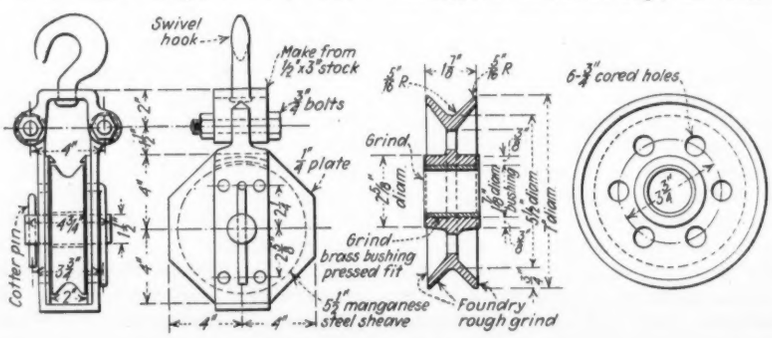


Fig. 67—5½-in. scraper block

steep enough in that part where the sides are flared to cause coarse pieces, which may be spilled by the scraper, to roll or slide down to the bottom. Otherwise there is danger that the scraper may get jammed. The early designs of slides usually had folding aprons in front that could be lowered until they rested on the floor; but under most conditions these have been found to be unnecessary, and the slide works satisfactorily if the bottom edge is 2½ in. above the rail. The first scraper load of dirt blocks the opening under the lip of the slide, and prevents the scraper from catching there. The height of the sides should be nearly as high as the scraper on the inclined portion, and about 25 per cent less on the horizontal part. Usually, the place where the greatest amount of spill occurs is just at the top of the slide. The clearance that the scraper should have between the sides should be not less than 2 in., and not more than 6 in. In other words, the slide for a scraper 42 in. wide should be not less than 44 in. nor more than 48 in. wide, inside measurement. Greater width is permissible, but it adds bulk and width to the slide without increasing efficiency.

The extension to the rear should clear the top of the car by 4 in. at the sides, and should have 6 to 8 in. greater clearance at the rear end along the center line, so that the dirt in the car will not be caught by it when the car is pushed out from underneath. This necessitates raising the rear sheave, and increases the over-all height of the slide. In drifts, where the slide must pass under the lips of chutes in going from one part of a level to another, this may be a serious drawback, although in operation to have a high rear sheave is advantageous because it reduces the wear of the rope on the plates at the top of the incline. The difficulty is overcome by mounting the sheave on a hinged bracket that can be folded back out of the way when passing under chutes. Standard practice is now to design slides so that they will have 5-ft. clearance over the track, and will pass under a 6-ft. chute. The rear sheave should be of the same size, shape, and material as that in the tail-block; but a larger pin may be necessary, because the rope pull is larger than is normal on the back haul.

A wet rope covered with fine quartz particles is a very effective cutting agent, and the slide should therefore be designed so that the pulling rope rubs on the plates as little as possible. Nearly every experimenter has tried introducing a roller in the floor of the slide at the point where the rope rubs, but these are never successful, because they are quickly jammed by fine particles and fail to rotate. The most successful plan has been to place a renewable plate of manganese steel, or boiler plate covered with stellite, at the point of greatest wear.

To hold the slide in position when loading cars a clamp is attached to each rail between the wheels. If the rear sheave is fastened to an eyebolt in the back of the drift, as is sometimes done, instead of being supported on the rear end of the slide, additional rail clamps must be used. These should be tight against the wheels on both sides, as the pull is alternately toward the front and back.

A slide that combines most of the points mentioned is shown in Fig. 63. It is designed so that the sheave can be carried on the slide or be hung from an eyebolt in the back, as desired, and is constructed in such a way that it can be easily sectionalized for transportation through small openings.

For ordinary work in drifts up to 9x9 ft. in cross-section, a 42-in. scraper, mounted on a 48-in. slide and pulled by a 15-hp. hoist, is standard equipment; but larger slides using 48- and 56-in. scrapers and 25-hp. hoists are becoming more and more popular for larger sections.

Several schemes have been tried for loading complete trains without switching. The first consisted merely of extending the horizontal portion of the slide sufficiently to cover several cars, which were loaded successively. Almost all of the loading had to be done at the rear end of the slide, and this slowed down the operation almost as much as it gained in switching. Other schemes were tried, of which the most successful was one in which the inclined slide is mounted on the end of the car nearest the breast, and the hoist is mounted on the locomotive at the other end of the train. The sides of the cars are raised, plates overlap the opening between cars, and angle irons or rails extend lengthwise along the tops of the cars, so that the scraper may be dragged along the top of the train and fill the cars successively. Although this scheme has been fairly successful, and good results were obtained by it, its use has not been general, and it is not as popular as the Osana slide.

One of the more recent attempts to reduce the amount of switching required is a scraper-car developed and used by the Montreal Mining Company, at Montreal, Wis. A 320-cu.ft. gable-bottomed car is used, which has an inclined slide mounted at one end and a hoist at the other. Two outfits are used, one of which loads at the breast while the other is being trammed to the shaft. A logical development of this idea would be the use of a self-propelling lorry car, instead of a car and locomotive.

Stationary slides are often used in breast stopes and in places where they can be left in one position for several weeks at a time. They vary in design from a fairly flat, open-ended stope-chute to a steel framework of the type shown in Fig. 64a. This particular slide, adjustable for height, may be used for end loading, as shown, or the cars may pass under the slide at right angles to its axis, and be loaded from the side. For end loading, the hoist is set up on the floor of the stope beside the track a little out of line with the slide, and the

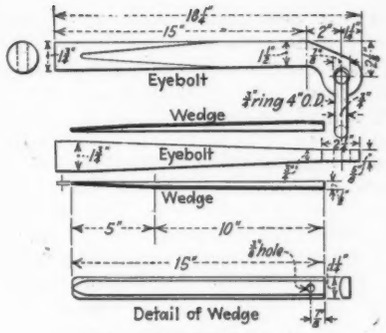


Fig. 68a—Eyebolt for 15-hp. scraper hoist

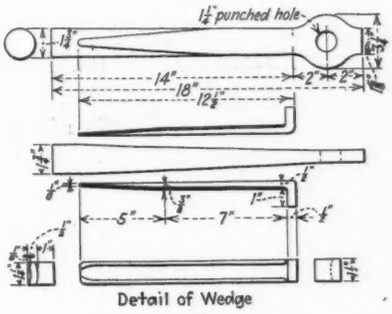


Fig. 68b—Eyebolt for 25-hp. scraper hoist

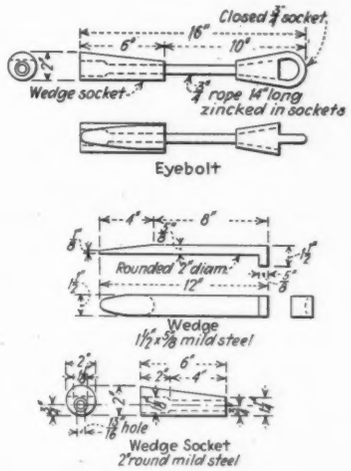


Fig. 68c—Flexible eyebolt for scraper

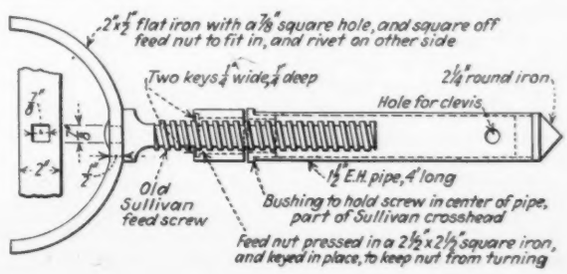

flat iron with square hole, and square off
feed nut to fit in, and rivet on other side
Two keys wide, deep
round iron
Hole for clevis
E.H. pipe, long
Old Sullivan feed screw
Bushing to hold screw in center of pipe, part of Sullivan crosshead
Feed nut pressed in a square iron, and keyed in place, to keep nut from turning

Fig. 68d—Scraper boom, made in part from scrapped piston drill parts

ropes pass through blocks hung from the back of the stope or from a crossbeam above the rear end of the slide. Side loading is the ideal arrangement when it can be used, because it reduces switching delays to a minimum. However, delays caused by switching cars are not as detrimental when scraper slides are used as with other types of loader, because the time taken in switching can be utilized in dragging ore up on the slide ready to be dumped, and in dragging it close to the foot of the slide, where it can be loaded quickly.

In Fig. 64b is shown a stationary slide used by the Montreal Mining Company for filling cars on the main level. Water in the ore passes caused so much trouble that chutes were dispensed with entirely, and the ore was allowed to run out on the floor of the crosscut, where it was picked up by a scraper and loaded into cars by means of a slide at considerable saving in expense.

The first scraper hoists were nearly all run by compressed air. In some districts this is still the general custom, notably on the Rand, in South Africa, where compressed air is used because of its beneficial effect on ventilation. In most places, however, electricity has replaced air as motive power. The cost of power transmitted by compressed air is approximately two and a half times that of electric power; and the large volume of air required by compressed-air hoists nearly always necessitates additional pipe lines and air compressors, which are more expensive than cables and transformers. Consequently, electric hoists in most places have superseded those driven by compressed air.

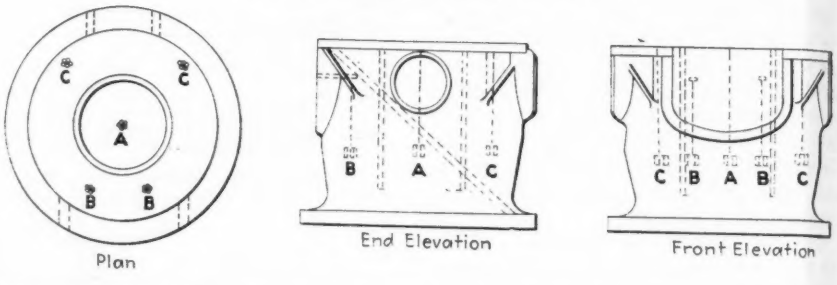
At first the smaller electric hoists were driven by direct-current motors, taking their current from the trolley circuit, but this tended to overload the trolley system, and the trend is now toward alternating current. Direct current is ordinarily supplied at 250 volts, and shunt-wound or compound-wound motors are used. Alternating current is normally 3-phase, 60 cycles, at 220 or 440 volts in the United States; 50 cycles and 550 volts are more common elsewhere. Little if any difficulty has been experienced in using 440- and 550-volt current in scraper work, and for these higher voltages the size of the motor and of the transmission cables is much smaller than for 220-volt current. Hoists equipped with friction clutches can use shunt-wound direct-current, or squirrel-cage induction motors; but those with jaw clutches require compound direct current or wound-rotor induction motors, which are much more expensive.

Cables in shafts should be armored and self-supporting; but those used on the levels need not be self-supporting, and can be carried on $\frac{3}{8}$-in. galvanized-strand "messenger wire," the attachment being made with tinned sheet-iron clips. The cost is little more than half that of self-supporting cable. If the distance between source of power and point of distribution is long, to take power underground at 2,200 volts and to transform it down to 440 volts near the point of consumption is an economical procedure.

A substantial junction box and switch at the shaft should be provided, where connection is made with the shaft cable and at points where more than one lead diverges from the main line. Single branches and connections at the motor are best made with Arktite, or similar, plugs and sockets, so constructed that they can be pulled without danger of arcing. For 3-phase current they usually have four poles, the fourth being for a ground wire.

The connections that have been described are also made in such a form that they can be screwed together like pipe unions, and as such are convenient for making temporary cable extensions. When these connections are used as unions or junction boxes care must be exercised, however, that the plug is not at the live end of the circuit, unless the contacts have been reversed so that they are not exposed.

Demolishing a 22-Ton Casting For Foundry Iron



DEMOLITION of the main frame from a worn-out 30-in. crusher, to provide foundry scrap, was undertaken recently at the property of the Bwana M'Kubwa Copper Mining Company, in Northern Rhodesia. Earlier efforts to break up crusher parts had not been entirely successful. Substantial quantities of explosive, for instance, had been used in "mud blasting," and much shop time had been consumed in the close drilling of large fragments that were subsequently broken with sledge hammers. Such methods proved inefficient and expensive.

The aforesaid cast-iron main frame weighed 22 tons, and was made of a particularly good quality of gray iron; as scrap it was worth about £90 to the foundry. An engineer of African Explosives & Industries, Ltd., successfully broke up the frame by hydraulic blasting, at a total cost of about 13 shillings. The accompanying sketch shows the main frame casting and the blasting charges.

Inasmuch as the crusher frame was divided into three separate compartments, electric blasting was employed to insure simultaneous explosion of the separate charges in each compartment. The casting was inverted to facilitate retention of water in the three compartments. All the orifices were closed up and caulked with tar and oakum; the vessel was then filled with water. A packet of ten sticks of $\frac{7}{8}$-in. 60 per cent gelignite was suspended at *A*, two-thirds of the way from the top, in the center of the compartment formed by the steel bushing. Two similar charges were suspended at *C* in the compartment formed by the diagonal partition or discharge floor of the crusher. Two charges of six sticks each were suspended at *B* in the other compartment formed by the discharge floor. One of the latter charges failed to explode, so the casting was actually broken by a total charge of 36 sticks of $\frac{7}{8}$-in. 60 per cent gelignite. The charges were wired in series, and care was exercised to make certain that the leads were carefully separated, as a short circuit might have occurred if the wires suspended in water had touched each other and the insulation wore thin at the point of contact.

One of the important advantages of this form of demolition is that, although the pieces are broken into comparatively small fragments, they are not thrown any great distance. The furthest outlying fragment of the shattered crusher frame was found at a distance of 47 yd. from the site of the blast; this piece measured about 2½x10x18 in. Most of the iron in this casting was 2½ to 3 in. thick. The force of the explosion, however, ruptured metal at points of reenforcement that was as much as 8 in. thick.

Surveying Diamond-Drill Holes

W. Rex Storms

TO BE OF ANY PRACTICAL VALUE, all diamond-drill holes over 300 ft. in length, and especially those of small EX diameter, should be surveyed. Numerous instances have been known where ore was found in a long, unsurveyed hole, but upon drifting to the supposed spot no ore was found. This failure occurred because the hole had swerved from its initial course. Thousands of dollars may be spent in drilling long holes, only to have them prove to be valueless when not surveyed. Certain holes that have been drilled for drainage, electric conduits, and other purposes have wandered off their initial course, and failure to survey and locate them has led to costs that proved much greater than was originally expected.

Until recently the only method commonly used for surveying small diamond-drill holes such as the AX or EX, which are usually used in prospecting for ore, was with the Maas patent drill-hole compass. This statement does not include large holes such as are used in oil-field drilling. The Maas compass consists essentially of two distinct parts: a compass needle suspended in a gelatine solution for obtaining the bearing of the hole, and a rubber-stoppered glass tube that contains dilute hydrofluoric acid for determining the dip. Usually both are included in one glass tube about 6 in. in length by 1½ in. in diameter. A rubber stopper is placed near the center of the tube, and the upper end is filled with a hot gelatine solution. The Maas compass itself consists of a magnetic compass needle suspended beneath a small cork float in such a manner that the needle is always horizontal. The lower end of the tube is partly filled with a dilute hydrofluoric solution, and both ends are tightly plugged with rubber stoppers. The sealed tube is now placed within a specially constructed brass or bronze case, which is screwed on the end of two brass drill rods. Insulation of the compass from the ordinary steel rods is provided by the brass rods. The outfit is then attached to the regular string of drill rods and lowered to the desired position. In a hole of ordinary rock temperature the gelatine will soon solidify and seal the compass needle in a fixed position at that point. This method gives the desired bearing of the hole, and the hydrofluoric acid etches a line on the inside of the tube, thus giving the dip. After taking the desired number of readings at various depths in the hole, the tubes are taken to the laboratory and the bearing and dip read on a goniometer. A goniometer, with dip tube only, is shown in Fig. 1. Because of capillary action within the tube a chart is furnished to correct for this action.

In deep holes a small thermos bottle is used for the hot gelatine and compass needle, the glass tube with acid for the dip being placed at the end of this bottle. Both tube and bottle are placed within the brass case as before. The thermos bottle keeps the gelatine from solidifying before the instrument reaches the desired depth. Agar-agar is also used to replace the gelatine in hot holes, as it will solidify at a much higher temperature. I have had no personal experience with this type of surveying instrument, but have used the acid and glass tubes extensively for obtaining drill-hole dips.

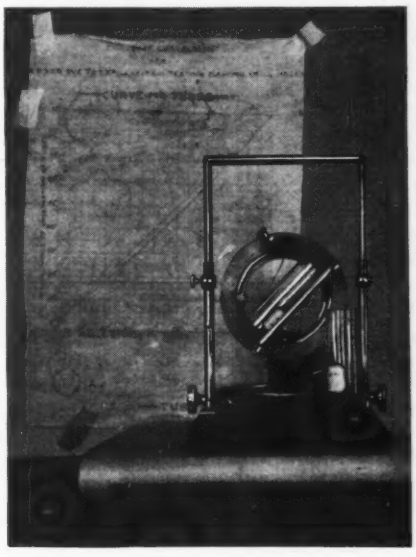
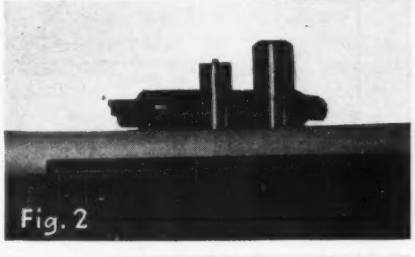
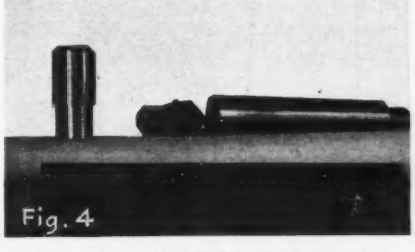


Fig. 1—Goniometer, tubes, and chart used in measuring borehole dip





Figs. 2 and 4—Wright instrument

A later method of surveying small boreholes is by use of the Wright borehole surveying instrument, shown in Figs. 2 and 4. An earlier type of this instrument was described in the *Engineering and Mining Journal* for July 14, 1928, under the title of "Borehole Surveying at the Homestake," by Lawrence B. Wright, chief geologist, Homestake Mining Company, Lead, S. D., who was one of the inventors. Because the drill holes at the Homestake wandered from their intended course in the steeply dipping, schistose, Pre-Cambrian rocks, the need of an instrument was keenly felt. Such an instrument was finally developed by Arthur B. Yates, geologist; A. A. Lease, expert photographer; and Mr. Wright. They wanted an instrument which did not use any liquid substance, light bulbs, timing devices, or wires leading into the hole, and, after much experimenting, they developed one.

In the original Wright instrument, a pendulously mounted case, with the bottom weighted to keep it in an upright position, and with a compass needle inside, was mounted within a small cylinder. The compass was attached to a cradle pivoted to this cylindrical shell in such a manner that the compass needle always remained in a horizontal position regardless of the attitude of the shell. The compass needle had radioactive material on each end, with the larger amount on the north-seeking end. A small piece of photographic film was mounted on a glass cover on top of the compass. The tube was placed within a specially constructed, waterproof, brass case, and the whole was lowered into the drill hole at the end of a string of brass rods. It was allowed to remain within the hole at the desired point for 15 to 25 min.; then it was drawn out, and the film removed and developed. The position of the compass needle was shown photographed on the film, making calculation of the bearing possible. For determining the dip of the hole, the compass box had a horizontal line of radioactive material around it, and the photographic film was placed around the inside of the cylindrical shell. When the instrument was allowed to remain in one position in the hole for 15 to 25 min. the horizontal line on the compass case was photographed on the film. By this means a curve instead of a straight line was obtained, as the distance from the radioactive material to the film varied. A special set of curves having known

dips had to be used to read the dip, the photograph on the film being compared with these curves.

The latest Wright instrument, shown in Figs. 2 and 4, consists of a small, square box swung from a frame, with the axis of the bearings parallel to the hole. This box, which will always remain parallel to the hole and in a vertical position, can be seen in Fig. 2. Within this box is swung a tiny box containing the compass needle. The axis of the bearings of this compass box is at right angles to the bearings of the first box; therefore the compass is always in a horizontal position. This frame fits into slots cut inside the main brass container. Fig. 4 shows the instrument removed from the brass case and so tilted that it is resting on one end of the frame. The axis, which is normally parallel to the main container

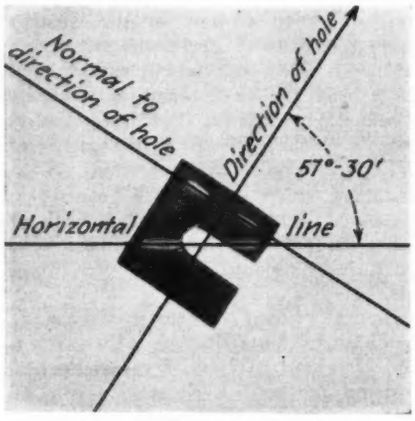


Fig. 6—Photograph showing dip of hole

and therefore to the drill hole, is vertical. A dip film is placed on each inner side of the outer box, and consequently these films are always parallel to the hole, one being on each side of the compass box. The compass box itself has a horizontal line of radioactive matter on each outer side. When the instrument is allowed to rest in one position for 15 to 25 min., the films take a photograph of the horizontal radioactive matter. As the films are always parallel to the hole, the dip is obtained from them directly and not from a set of curves, as was done with the original instrument. Photographic reproductions of these dip films are shown in Fig. 3. The two shorter white lines on the dip film are normal to the hole; the longer white lines are made by the horizontal lines on the compass box. The film is cut in such a manner that it will slip down over the bearings of the compass box.

The method of calculating the dip is indicated in Fig. 6. Care must be used in placing the films within the outer case. The two short vertical lines should always be in a known position, either toward the end of the drill hole or toward the collar. If this is not done, the surveyor will not know

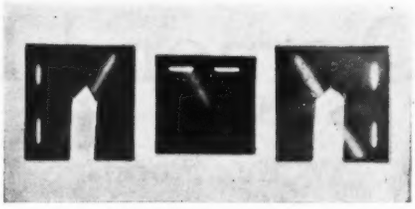


Fig. 3—A bearing photograph and two dip photographs, all made with the Wright instrument

whether his hole is going up or down, especially on a supposedly flat hole.

To obtain the magnetic bearing of the hole, a film is placed in a small holder, and the holder placed on top of the compass box, sensitized side down. In the original model this holder was made of glass, but it is now made of metal. The compass needle has radioactive material on both ends, the larger amount being on the north-seeking end. A photograph is made of this needle, similar to that made for the dip; from the developed film the magnetic bearing of the hole can be obtained. The middle picture in Fig. 3 is that of the developed film, the two short lines showing the axis of the hole, and the long and short lines showing the needle position. Fig. 5 shows the method of obtaining the magnetic bearing. As with the dip films, care must be used so to place the film on top of the compass that the guide lines showing the direction of hole will always be in the same known position, either on the right or left side of the instrument when looking toward the end of the drill hole. In obtaining the magnetic bearing, the fact that the film holder itself has two short lines of radioactive matter on it must be borne in mind. Consequently the holder should be placed on top of the compass, so that these lines will always be on the same side.

For the actual surveying, the Wright instrument is loaded with films in a dark room, and then taken to the hole to be surveyed. Underground, the instrument may be loaded at the drill after extinguishing all lights, the final operation being completed with the aid of the light from a photographer's red lamp. Films are carried to the hole to be surveyed in a black-lined, lightproof box. An ordinary flashlight with red tissue paper placed between the bulb and lens may be substituted for a photographer's lamp. The films are cut in a dark room by the surveyor, or they may be purchased already cut to fit the instrument.

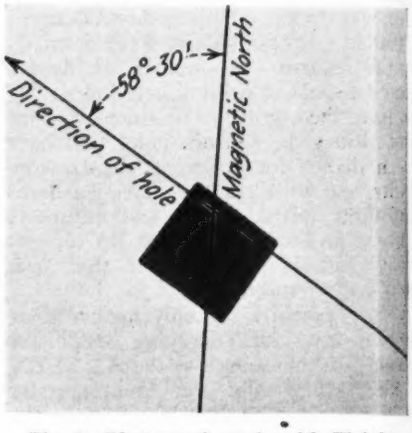


Fig. 5—Photograph made with Wright instrument showing bearing of a hole

After loading, the instrument is placed within its brass case, and the case then made watertight by carefully wicking or greasing its threads. Black envelopes for carrying the exposed films are readily made from the black paper used as wrapping on photographic plates. Use of a separate black envelope for each set of films will prevent them from becoming mixed when taken to the dark room for development.

After loading the instrument into the case, the latter is attached to a 20-ft.

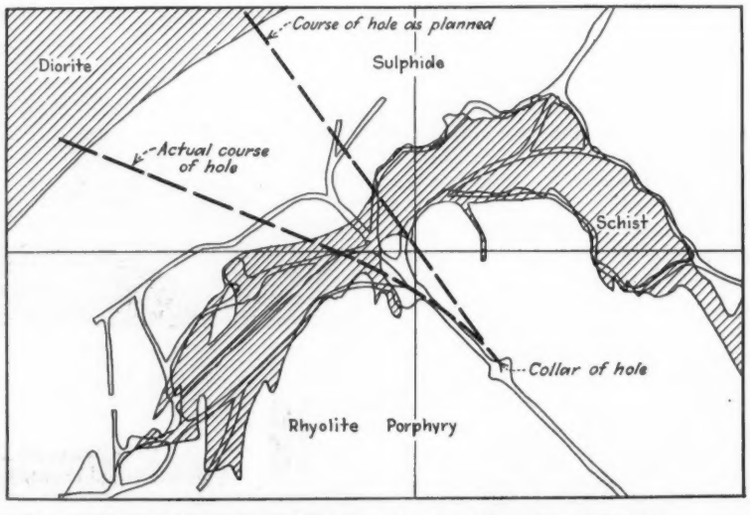


Fig. 7—Plan of mine level showing deviation in a drill hole
This hole was started with a bearing of N. 45° 53' W. and a dip of minus 30° 30'. The hole was not surveyed, but in later stoping operations it was found to be in the position indicated, with a bearing of N. 68° W. and a dip of minus 14° at a distance of 385 ft. A vertical section showing the hole is reproduced in Fig. 8

brass rod made up of two 10-ft. sections, which rod is then attached to the ordinary steel drill rods. Two lengths (20 ft.) of brass rods seem to be sufficient to insulate the compass from the steel rods. The rods are then lowered into the hole to the desired depth, and allowed to remain there for 20 min., in which period a satisfactory photograph is made. The instrument is withdrawn from the hole, the films are changed, and the rods again lowered. When the hole has been surveyed, the films are taken to a dark room and carefully developed. For determining the dips and bearings a protractor can be used, lines being drawn through the film as shown in Figs. 5 and 6.

The Wright borehole surveying instrument cannot be used for surveying holes drilled in formations containing magnetic matter, nor can it be used in holes that are cased with ordinary steel casing. Usually a 1,000-ft. hole should be surveyed at depths of 50, 100, 150, 200, 300, 400, 500, 700, and 900 ft. However, if possible it should be surveyed every 50 ft. to obtain definite bearings. My experience would indicate that most drill holes, when drilled in fairly uniform rock, do the greater part of their curving within the first 200 ft. Also, most of the deviation is in dip and not in bearing. When drilled across varying formations, however, both the dip and the bearing of a hole will vary according to the formation.

Data from the actual survey of a 395-ft. EX drill hole that was drilled upward, supposedly at 60 deg., through fairly uniform rock, are presented in Table I. The dip is usually read four times, twice with each film. As a rule, the bearing is read at least twice. The more readings taken the better, as the correct figure is difficult to obtain by reading the film only once.

Table I—Survey Data for an EX Drill Hole

Footage	Dip	Magnetic Bearing
10	57° 30′	N. 58° 30′ E.
50	56 37	N. 59 07 E.
100	51 00	N. 62 00 E.
150	55 30	N. 60 00 E.
200	49 15	N. 58 52 E.
250	46 52	N. 59 15 E.
300	42 22	N. 57 52 E.
350	40 30	N. 59 30 E.
395	39 07	N. 60 07 E.

Table II shows the time required to survey the aforesaid hole, the survey being made by the geologist, the driller, and the driller's helper, as soon as drilling was finished. Only one instrument was used, the geologist changing the film and the driller and helper putting down and pulling rods.

The column marked "Instrument Time" indicates the time required to load and unload the Wright instrument. "Down" is the time used for putting down the rods, being actual time from collar to desired position. "In" is the time the rods and the instrument were allowed to remain at the desired position. "Out" is the time of actual pulling. At 350 ft., the time the rods were in the hole was lengthened to 30 min. and the pulling time also lengthened by an unavoidable delay. The film for this 30-min. stay was good, showing that 20 to 30 min. is about the right length of time for the rods to be in the hole. The film used was slow film, however, and not speed film. Slow film can be handled in more light than a speedier film and does the work just as well. The time of 12 min. to load and unload the instrument at the 395-ft. distance was caused by the circumstance that one of the films was so large that it had to be cut down with a pocket knife to fit it into the film holder.

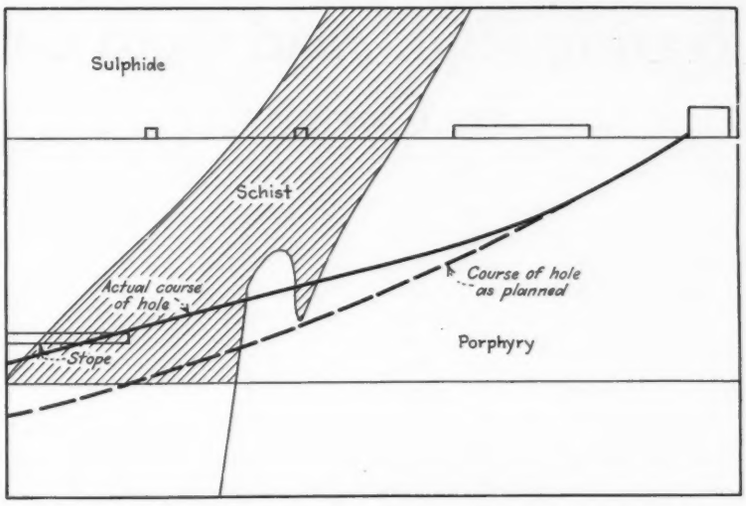


Fig. 8—Vertical section showing the diamond-drill hole indicated in the mine plan reproduced in Fig. 7

Table II—Time Required to Survey Drill Hole

Distance, Ft.	Instrument Time, Min.	Rod Time, Min. Down	In	Out	Total Min.
50	6	5	20	3	34
100	6	6	20	6	38
150	3	11	20	10	44
200	5	19	20	13	57
250	5	20	20	14	59
300	5	22	20	15	62
350	7	29	30	25	91
395	12	38	20	24	94

Geology of Pioche Ore Deposits

OREBODIES of the Pioche, Nev., and surrounding districts comprise both oxidized and unoxidized deposits. They are reviewed by Adolph Knopf in U. S. Geological Survey Professional Paper 171, recently issued. Oxidized deposits predominate, but during recent years the unoxidized deposits have yielded the most ore. The two classes differ greatly in appearance and composition. Unoxidized or primary deposits consist chiefly of silver-lead-zinc ores. As a result of oxidation the primary ores have been much changed, and the changes produced have been influenced by the nature of the country rock in which the ore deposits are inclosed. The oxidized deposits that are inclosed in quartzite are silver-lead ores, for the zinc they originally contained was largely eliminated during oxidation. Copper in commercial amount occurs with the other metals in some of the ores, as at Bristol, and tungsten in the form of wolframite is a minor constituent in others.

Manganese oxides, derived from the oxidation of manganiferous siderite, are common in the ore deposits in limestone. Some ores are valuable chiefly as manganese-iron fluxes, and the ore of the Jackrabbit mine contains so much manganese and is so low in silver, lead, and zinc that it is utilized as a manganese ore in iron smelting.

For descriptive purposes the ore deposits fall more conveniently into three groups—fissure veins in quartzite, mineralized granite porphyry, and replacement deposits in carbonate rocks. All of them appear to have been formed at the same time.

The Pioche region is greatly faulted, the faulting having occurred at several different times. That there were three epochs has been proved, and probably there were more. With reference to the ore deposits, pre-mineral, mineral, and post-mineral faults can be recognized. In the search for ore, the distinction between these three classes is likely to be of importance.

The displacement along the fissures occupied by the veins is generally small; in few does it exceed some tens of feet, and in others, as in the Mendha vein, no displacement has occurred. As in many other mining districts, the master faults, however, are not mineralized. Post-mineral movement has broken the continuity of some of the orebodies by closely spaced step faults. Movement probably has recurred along certain faults during two or even more epochs of faultings, according to the author of the monograph.

Treating High-Acid Gold Ore in Cyprus

C. A. Rose

A GOLD ORE DEPOSIT now being worked at Skouriotissa, Cyprus, has unique characteristics. The ore is a clay-like material weighing only about 75 lb. per cubic foot in place, and 55 lb. per cubic foot after mining. It contains about 18 per cent soluble salts, mostly sulphates of iron, and 15 to 18 per cent moisture. In appearance it is a dry, soft, crumbling material, with the peculiar property of becoming liquid slime when pressed between the fingers. For this reason and because it is very corrosive, it has been called Devil's Mud by the miners. The ore titrates 2.5 per cent free sulphuric acid.

Devil's Mud is adjacent to and partly underlying the outcrop of a large copper-pyrites deposit; apparently it originated from a portion that was oxidized by the atmosphere, dissolved by rain water, and washed away. The ore lies mostly in an irregular horizontal stratum 6 in. to 6 ft. thick, covering an area of about ten acres, although a nearly vertical vein exists where this soft material was evidently washed into a fault fissure.

Under the gold ore and under the present pyrites deposit is pillow lava—the country rock of Cyprus. In places this has been whitened and sericitized by the leaching of acid waters, from the former deposit of pyrites above, as it was corroded. Devil's Mud appears to be a mixture of clay residue from the pyrites and siliceous residue from the leached pillow lava. Together, they form a sponge-like substance that retains acidified iron salts derived from the pyrites. If the gold in Devil's Mud came from the pyrites that was formerly overlying it, and if this pyrites were no richer in gold than the portion occurring in the Skouriotissa mine, about 100 tons of pyrites must have been oxidized and washed away for each ton of Devil's Mud formed, and the pyrites deposit must have been about 100 ft. thick.

The gold in Devil's Mud is readily soluble in cyanide solution, and a plant is being constructed for treating the ore by that method. Before cyaniding, however, the acid and iron salts must be washed out. Grinding of this corrosive ore, which presented somewhat of a problem, will be done in a rubber-lined pebble mill, in circuit with an acid-proof Dorr classifier. The ground ore will be washed with a large excess of water, and settled in a Dorr thickener. The thickened pulp will be cyanided by agitation and washed by counter-current decantation, with final filtering of the tailing.

Eighty per cent of the gold in Devil's Mud can be recovered by flotation, but the concentration ratio is low. The gold cannot be amalgamated, nor concentrated by panning, which probably explains why this deposit was left by the ancients who worked the Skouriotissa pyrites deposit extensively, as evidenced by the enormous slag piles they produced. Apparently they had no means of knowing that Devil's Mud contained gold. Possibly, however, the deposit was originally much larger, and most of it was mined and used as siliceous flux in smelting copper pyrites. A slag pile of approximately a million tons, containing about 30 per cent silica, remains near the mine. The pyrites contain less than 1 per cent silica, and no silica deposits are now found locally. Therefore, apparently, the 300,000 tons of silica in the slag was imported or came from a deposit that was thus exhausted.

No historical evidence seems to be available that gold was ever produced in Cyprus, but the following method, described in Hofman's "Metallurgy of Copper," could have been used by the Romans to recover gold from copper matte produced by smelting Devil's Mud with copper ores at Skouriotissa:

"The Pearce Gold Separating Process, which was carried on in secret from the starting of the works of the Boston & Colorado Smelting Company, at Black Hawk, Colo., in 1874, until the closing of the Argo, Colo., plant in 1909, is based upon the strong affinity S has for Cu in comparison with Au, Ag, and other metals found in copper bottoms. If a granulated bottom is repeatedly fused with pyrite in right proportions, Cu, Ag, and the other metals will be 'stripped'—i.e., taken up by the matte that is formed—while impure Au will remain behind to be purified by niter fusions. The same principle of working has been developed independently by Roessler for the refining of base jeweler bars. At Argo the practice was as follows:

"A charge of 6,400 lb. bottom (0.4 to 0.5 per cent Au) is purified by melting in 8 hours on the the brick hearth of a reverberatory furnace in an oxidizing atmosphere, the temperature being raised toward the end with doors and ports in fire-bridge closed to render the whole thoroughly fluid; the fluid charge is rabbled, the slag skimmed, and the alloy tapped at the side, conveyed through a cast-iron launder into a tank of cold water; the stream of metal falling first onto a spruce pole to spray it before it strikes water. The granules, weighing about 4,400 lb., are caught in a perforated pan, removed and dried. From 14 to 16 charges of bottom furnish enough material for from 22 to 30 charges for the first stripping. A mixture of 220 lb. granules and 1,500 lb. pyrite is melted in a smaller reverberatory furnace; the slag is skimmed; the matte together with alloy tapped into communicating sand molds; the alloy weighs from 625 to 650 lb. When enough alloy has been accumulated, it is melted in a reverberatory furnace as was the crude bottom, skimmed, tapped, and granulated; the granules contain from 2 to 3 per cent Au. The first stripping is followed by a second, giving an alloy with 25 per cent Au, and this by a third which concentrates the Au to 45 per cent. The further refining is carried on in graphite crucibles; three strippings bring the Au to 900 fineness, when a final treatment with niter in clay crucibles removes more base metal and furnishes a gold bar of about 957 fineness. The mattes which are gold-bearing are returned to preceding operations as are other intermediary products."

Automatic Oiler for Hoisting Rope

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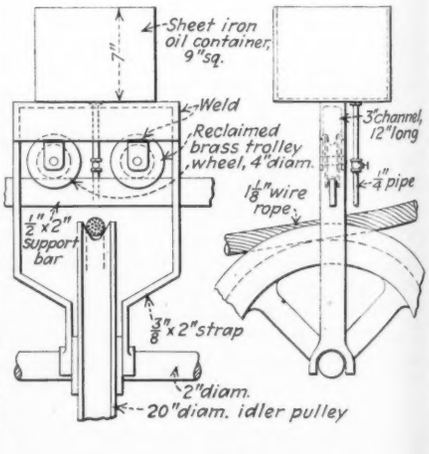
AN AUTOMATIC OILER for hoisting ropes that can be made in the blacksmith shop is shown in the accompanying sketch. This oiler has been in satisfactory operation on both cage and skip ropes at Armour No. 2 mine, near Crosby, Minn. The apparatus, guided horizontally by the idler pulley, is supported on two worn locomotive trolley wheels, which turn on a ½x2-in. bar attached to idler stand posts. A ¼-in. angle valve regulates the quantity of lubricant supplied to the rope. A light lubricant that has penetrating qualities is more desirable than a heavy mixture.



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Iron-Ore Mining and Milling At Scrub Oak—I

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An account of an operation in the Highlands of New Jersey made possible by the amenability of the comparatively low-grade magnetite ore to concentration by magnetic separation, despite the fact that about one-fifth of the total iron is in the form of non-magnetic martite

THE Scrub Oak mine of the Alan Wood Mining Company is in the Highlands of New Jersey, 3 miles west of Dover and about 40 miles west of New York City. Iron ore produced is of bessemer grade. The property was opened in 1856, and about 25,000 tons of ore was produced. Mining then stopped, as the ore contained only 33 per cent iron and was too lean for furnace use. In those days the only means of concentrating iron ore was by hand sorting or cobbing, and on finely disseminated ores, such as that at Scrub Oak, hand cobbing was not successful. Scrub Oak ore must be crushed to 8 mesh to unlock enough mineral to permit the production of a satisfactory concentrate and obtain a recovery of approximately 92 per cent.[1]

In 1917, the Wharton Steel Company reopened the mine to obtain a supply of bessemer ore for its furnaces at Wharton, N. J. The ore was treated by dry magnetic separation and wet gravity concentration, Ball-Norton drums and belts being used for the former, and pulsator jigs and shaking tables for the latter. This dual treatment was necessary, because 18 per cent of the iron in the ore was martite, a black, hard specular hematite, the Scrub Oak ore differing in that way from most other ores in the district, which as a rule are wholly magnetite. Production stopped when the Wharton furnaces were closed down in 1923.

The mine remained idle until the latter part of 1929, when it was taken over and placed in operation by the Alan Wood Steel Company, Conshohocken, Pa., to supply bessemer ore for the company's Swedeland furnaces.

The Highlands of New Jersey are formed mainly of highly metamorphosed rocks of Pre-Cambrian age, which consist of gneisses, gneissoid granites and schists, more or less banded, with some marble or crystallized limestone and intrusive igneous members. In this region the strata have been greatly deformed and are now complexly folded and faulted. The ridges trend northeast and southwest. The country is fairly rugged, the crests of the hills being, in general, 200 to 300 ft. and in some instances as much as 600 ft. above the intervening valleys. Geology of the Highlands and their iron-ore deposits is discussed in detail in the Reports of the Geological Survey of New Jersey. (William S. Bayley: "Iron Mines and Mining in New Jersey." Vol. 7. Geological Survey of New Jersey, Trenton, N. J., 1910.)

Agreement is general that the iron ores are of magmatic origin and were produced from a reservoir at great depth, which furnished the material from which were also formed the gneisses and pegmatites comprising the major part of the Pre-Cambrian rocks of the Highlands.

Occurrence of the bodies of magnetite appears to be independent of the variety of the country rock, and no structural features or conditions peculiar to the ore ranges have been noted. Persistence of the veins or lenses of magnetite in New Jersey is remarkable, and, so far as is known, no instance is recorded of any deposit having bottomed. Many deposits have been worked to depths, as measured along the pitch, of 4,000 to 10,000 ft., the bottom still showing ore. The statement appears warranted therefore that once the size, shape, and mineral content of any oreshoot is determined, these characteristics will continue to any attainable depth, with a slight tendency to increase in size and mineral content as they approach their origin or source.

The magnetite is widely distributed in the several varieties of gneiss, pegmatite, and limestone, as thin lenses or sheets and sometimes as thicker lenses or pipes which strike and dip conformably with the country rocks. Prevailing strike is northeast and the dip southeast, with a few exceptions where the dip is northwest. The oreshoots pitch downward to the northeast at angles of 10 to 60 deg., as do the inclosing rocks.

In the formations mentioned in the foregoing the oreshoots may be compared to a board or plank that has been shoved endwise into the earth with the narrow edge up, then partly turned sideways, and then tilted downward at one end. The thickness of the plank represents the thickness, and the side or face the height, of the oreshoot. That part of the plank intersected by the ground surface represents the outcrop, and the rocks at the sides or faces of the plank are the footwall and hanging wall. The rock at the bottom edge of the plank is called the bottom rock and that at the upper edge, the top, or cap rock. When the plank is turned partly sideways, it has a dip similar to the dip of the oreshoot. By tilting the end of the plank downward, it then has a pitch similar to that of the oreshoots.

Minable oreshoots range from 6 to 100 ft. wide, as measured from footwall to hanging wall. Their heights,

[1]—The concentration of this ore will be discussed in Parts III and IV of this article.

General view of Scrub Oak surface plant, near Dover, N. J.

as measured on the dip and normal to the long axis of the oreshoot, vary from 50 to 3,000 ft. Length along the pitch has never been determined, as the end of no oreshoot has been found by mining, although some of the orebodies have been worked to lengths of 10,000 ft. The grade of the ore as mined varies from 30 to 60 per cent iron. Gangue minerals are chiefly silica, alumina, magnesia, lime, and phosphorus. As a rule, the ores high in iron are also high in phosphorus and the lean ores are low in phosphorus.

Average dip of the oreshoots is 55 deg. The footwall and hanging walls are firm and stand well. Usually the footwall is smooth, and shrinkage stoping can be used. Where a flat pitch or dip, or both, make this method impracticable, underhand stoping is used with or without sublevels. With the underhand method, at least part of the ore must often be scraped to loading chutes. No timbering is required except at hoisting shafts, where it is more a means of support for equipment than for the shaft walls.

The oreshoots are in well-defined ranges, extending from the Delaware River, across northern New Jersey, to the New York State line at Ringwood and beyond into New York State in a general northeasterly direction. The grade of ore found in any range is approximately the same; that is, some of the ranges are known as high-phosphorus lines with high iron content, and others are noted as containing very little phosphorus and as being low in iron. Of significance in this connection is the fact that in the gneisses, the ores associated with pegmatites are generally high in iron and phosphorus, and the scarcity or absence of pegmatite denotes low iron and phosphorus content. At several mines are parallel shoots worked from one shaft, and in some places several oreshoots are found, one above the other, but lying in the same plane, which are also worked from one shaft.

All oreshoots outcrop, and the outcrops can be found by cleaning off the surface sand and gravel, except where they have been displaced by faulting. From the outcrop, the width, height, dip, and pitch of oreshoots, and the iron content, can be determined. These data seemingly hold good, however deep the oreshoot may be mined. The pitch of the shoots is remarkably regular; inclined shafts, partly in ore and partly in the bottom rock, have been driven along the bottom for distances of 2,500 to 6,000 ft., and practically no deviation of ore and rock from a straight line has been found, unless the shoot has been displaced by faulting. This is particularly true of shoots of high iron content. In the lean oreshoots, contact of the bottom rock and ore is not so well defined, the shoot gradually diminishes in ore content until it becomes non-commercial, and at the same time displays a tendency to "finger out" into several stringers. Whether the top and bottom rocks of the leaner veins, on which work is now commencing, will be as easily followed and recognized as were those of the richer ones remains to be seen.

Post-mineral faulting has not been found to be extensive, and the oreshoots are mostly continuous without major displacements. Generally faults do not seriously interfere with mining, because most of the cross faults have a horizontal displacement of less than 100 ft., and the strike faults have a vertical displacement of 20 ft. or less. The cross faults normally strike northwestward and about at right angles to the strike of the oreshoots. They dip mostly to the southwestward, at angles between 45 and 90 deg.; and the majority are normal faults. A number of faults strike nearly parallel with the oreshoots and dip northwestward. Their dip is opposite to that of the shoots and at about the same angle as that of the shoot they cut. These are reverse faults, with displacement, as measured on the fault plane, about equal to the distance between footwall and hanging wall of the oreshoot they cut.

Discovery of "blind" shoots, or ore that does not outcrop, is thought unlikely, except where the bodies have been displaced by faulting and cannot be located by surface prospecting. That is, all the magnetite oreshoots are held to have reached the surface before faulting occurred, and no shoots that did not once outcrop are expected to be discovered in depths reached by present-day mining.

At the Scrub Oak mine the ore deposit consists of a single lens or shoot of iron ore, about 25 ft. thick and 900 ft. high as measured on the dip and normal to the pitch. The outcrop is 2,500 ft. long, although the southerly tip has been removed by faulting. Strike is N. 33 deg. E., the dip 55 deg. S.E. The pitch is to the northeast at about 18 deg. Pitch angle has not yet been definitely determined, as mining has not progressed far enough to uncover the top and bottom rocks. The walls of the deposit are hard, close-grained granitoid gneiss, which stands well in mining. At No. 1 shaft, a normal fault displaces the ore 40 ft. horizontally to the southeast, and the presence of a larger fault 550 ft. northeast of the small fault is indicated. The larger fault is thought to have displaced the oreshoot 400 ft. horizontally to the southeast and to be a normal fault. At any rate, by magnetic survey, diamond drilling, and surface pits, an orebody known as the Erb has been located 400 ft. southeast of the Scrub Oak shoot, and has the same general characteristics as the latter. Both ores assay 33 per cent iron and 0.07 per cent phosphorus, are finely crystalline, and composed of a mixture of magnetite and the non-magnetic martite. Accessory minerals in both veins are chiefly silica, with small amounts of alumina, lime, magnesia, pyrolusite, and apatite.

Prospecting was originally done by surface trenching, test pits, and sampling of the outcrop and old workings, before mining was started in 1917. A general magnetic survey of the surface was then made. This outlined the location of the oreshoot and served as a guide in planning the number and location of the diamond-drill holes which followed. The drill cores were $1\frac{1}{8}$ in. diameter and core recovery was approximately 90 per cent. About 27,000 ft. of drilling was done from the surface to intersections with the vein, at a contract cost of $3.60 per foot for holes up to 1,500 ft. deep. Subsequently, a small Sullivan drill has been operated underground on company account, to guide development work throughout the mine and particularly in the faulted areas where horizontal holes as long as 400 to 500 ft. are necessary. This work costs about $2 per foot. Shoot width is determined at intervals of from 50 to 100 ft. on main levels by hammer drills and sectional steel with detachable bits, where lengths of hole do not exceed 40 ft.

Estimate of ore reserves at Scrub Oak presents little difficulty, inasmuch as there is only one class of ore and the oreshoot maintains its size and shape with great regularity. Diamond-drill cores are split for assay; and, where core recovery is low, part of the sludge is caught and assayed. These are combined to obtain the average assay of each hole. Sludge samples taken from hammer-drill holes are settled in baffled containers, dried, and assayed. Drifts, raises, and other underground openings are channel-sampled by hand. Once the grade of the oreshoot from bottom to top rock is determined, further sampling is not necessary.

Tonnage of ore reserves was computed by determining the superficial area of the oreshoot on its dip plane, as proved by diamond drilling, and multiplying this area by the average thickness as determined from diamond-drill cores, surface pits, outcrops, and by mining. The tonnage factor is 10.5 cu.ft. to the gross ton. Sufficient reserves have been indicated to provide for at least fifteen years' operation, at the rate of 500,000 tons of crude ore per year.

Up to 1912, iron ore was mined in New Jersey by underhand stoping. Inclined shafts were sunk on the footwall in ore and at intervals of about 400 ft. along the outcrop. Hand drilling and hand tramming were universally used and broken ore was hoisted in buckets by horse whims and small steam hoists. Underground stoping proved dangerous and expensive; but, notwithstanding its evident drawbacks, large tonnages were removed from many of the older mines. Use of this method made it necessary to drive levels at

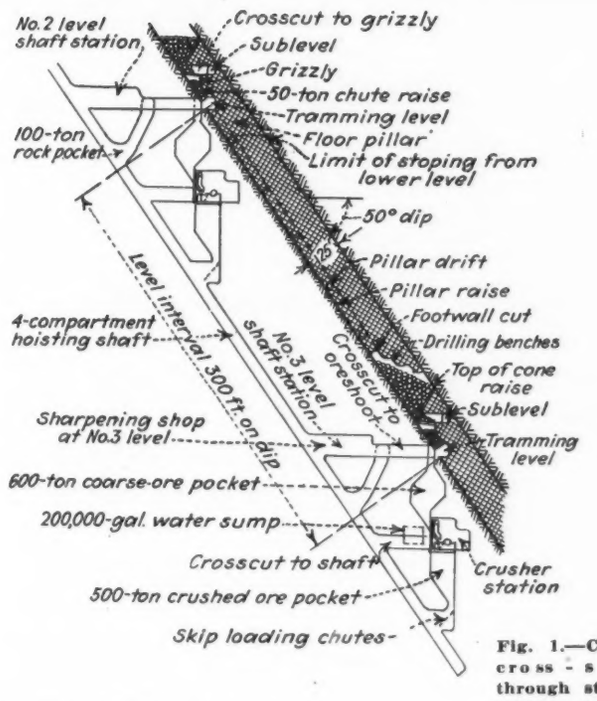

Fig. 1.—Composite cross-section through stope and hoisting shaft, Scrub Oak mine

close intervals, 50 to 100 ft. apart vertically, and to leave substantial floor pillars for overhead protection and wall support, thereby reducing total extraction considerably. Unrecoverable ore in pillars amounted to 30 to 40 per cent.

In 1912, shrinkage stoping with electric haulage was introduced by one of the writers of this article, then superintendent of the Mount Hope mine. This method quickly became general practice at New Jersey mines (H. M. Roche and J. C. Stoddard: "The Mount Hope Mine." *Iron Trade Review,* July 22, 1915); and in 1917, when the Scrub Oak property was reopened by the Wharton company, the mine was developed for shrinkage stoping, amplified to include sublevels with a grizzly at each stope chute.

Today, the Scrub Oak mine is worked through a four-compartment shaft, sunk in rock 100 ft. back of the footwall side of the oreshoot, as shown in Fig. 1. The shaft is near a point equidistant from either end of the outcrop. From the shaft, crosscuts in rock are driven to the oreshoot, and haulage levels are then driven in ore on the footwall until the bottom and top rocks respectively are reached. At intervals of 40 ft. along haulage levels, grizzly raises are driven on the footwall to a height of 25 ft. above the level floor, and at their tops bulldozing chambers are cut out, and these chambers connected by short crosscuts to a sublevel. Grizzlies are built in the bulldozing chambers, and directly above them are draw raises connecting with the stope bottom.

Shrinkage stoping is conducted with the strike. Stopes are 200 ft. long on the first level, 280 on the second, and 320 on the third, with 40-ft. pillars between. Access to stopes is through raises in the stope pillars, and these raises extend to the level above to afford ventilation in the stopes. Three levels have been opened, with the interval increasing from 232 ft. for the first to 278 for the second and 300 for the third. Use of comparatively short stopes allows better control of stope dimensions and results in a minimum of dilution. The ore was structurally weak above the first level, but at lower levels became sufficiently strong to allow increased lengths and heights in stopes, with consequent reduction in development work. The ore is sandy, does not pack, and high recovery of that broken in stopes is obtained.

No. 1 or main hoisting shaft is an inclined four-compartment shaft, sunk in the footwall about 100 ft. from the ore (Fig. 1). Compartments are 6 ft. 2 in. in the clear, and the shaft is inclined at 55 deg. from the horizontal, or at the dip angle of the shoot. The shaft is timbered in the usual manner, except that the first 90 ft. is lined with reinforced concrete. Total depth is 1,010 ft., measured on the dip. Ore is hoisted in two compartments, rock, men and supplies are handled in the third, and in the fourth are a ladder-way, pipe lines, and electric cables. Skip tracks are of 90-lb. standard rail clamped to wood ties. Ore skips hold 5 tons. Present rope speed is 800 ft. per minute. By installing an additional hoist motor and changing the gear ratio, the rope speed can be increased to 1,500 ft.

The ore hoist is a double-drum Wellman-Seaver-Morgan unit, with drums of 72-in. diameter and 60-in. face, and is driven by a 350-hp., 2,200-volt, 3-phase, 60-cycle motor. The capacity of the hoist is 12,000 lb. of ore.

The man hoist is a single-drum Wellman-Seaver-Morgan hoist, capable of handling 30 men, 5,400 lb., with cage weighing 5,000 lb. and 1,800 ft. of 1-in. rope, weighing 2,840 lb. Hoisting speed is 437 ft. per minute under full load. The drum is of 60-in. diameter and 60-in. face. Brakes are the post type and operated by hand. A Lilly safety governor is used and a solenoid brake is automatically applied in case the current fails. The hoist is driven by a 140-hp. d.-c. motor.

Three levels have been opened, and on each a shaft station, 30 ft. wide, 50 ft. long and 15 ft. high, is cut out, and an 8x10-ft. crosscut run to the oreshoot. From the crosscuts, haulageways are driven to right and left in the ore. Fig. 2 is a plan of the third level station.

At each station a 600-ton ore pocket has been cut out under the main haulage level and in the rock close to the footwall. Immediately below this pocket is a crusher station accommodating a 30x42-in. jaw crusher with motor drive and feeder. Below the crusher is a 500-ton crushed-ore bin, from which the ore passes to a measuring chute and to the skips. The maximum size of ore dumped into the upper pocket is approximately 1.5x2x3 ft. The crusher is set to 6 in., thus allowing hammer heads, broken rails, and other steel to pass.

The second-level ore bins made considerable water, because they were close to or in a fault zone. Although no large streams were met, water entered from all sides through fine cracks, wetting the ore. This caused much difficulty in screening in the mill and necessitated running the dryer full blast continually. Various methods of overcoming this trouble were discussed, the final decision being to grout the bin walls by means of the Francois method of cementation. Results were satisfactory.

Rock broken in development is not mixed with ore but is trammed and hoisted separately, where possible. The man hoist is used for hoisting rock.

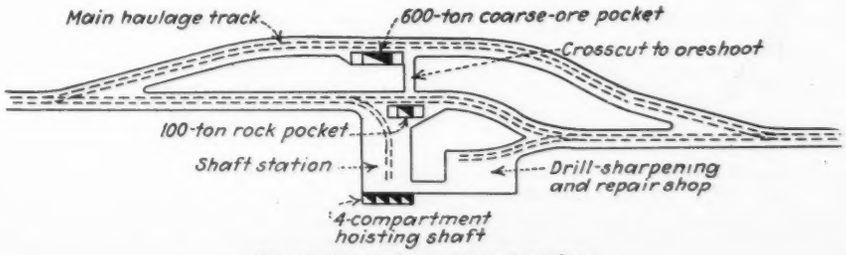

Fig. 2—Plan of No. 3 level station

On each level, a bypass drift is run around the ore dump and connects with the main haulage level on each side of the shaft station, as shown in Fig. 2. Underneath the bypass drift is a rock pocket from which rock is loaded into the man skip. With this arrangement, no interruptions to ore tramming occur, and dilution of ore by development rock is avoided.

Development of levels is started after the completion of the shaft station, ore and rock pockets, skip-loading chutes and gates, and the crosscut from shaft station to the oreshoot. Tramming drifts are 8½ ft. high by 11 ft. wide, and are driven mainly in ore on the footwall.

Drill rounds in headings consist of 22 to 26 holes, placed as shown in Fig. 3. A wedge cut is used and holes are drilled with 8-ft. steel. The average round breaks 6.5 ft., yielding 57 tons of ore. Drilling and blasting are completed in each heading in one shift by two drillers and one helper. A Nordberg-Butler shovel is used. It is fast and stands up well in handling the heavy ore, which runs about 16 cu.ft. per gross ton. Both headings are drilled and blasted on one shift and mucked on the following shift by one shovel loader.

Equipment used in headings consists of the shovel, a storage-battery locomotive, and a 5-ton Granby car. This equipment is handled by two men, a shovel operator and a locomotive operator. Track extensions are made in 5-ft. lengths and by the shovel crew. In moving from heading to heading, the shovel is coupled to the Granby car and both are shifted by the locomotive.

Before the shovel was installed, four muckers and one locomotive operator were required per heading per shift. Using the shovel, only two men are needed to muck out two headings per shift. Eight men in two headings per shift are thus saved, which amounts to $3.16 per foot of drift. From this must be deducted the cost of operating the shovel, which is $0.60 per foot of drift, leaving a net saving by the shovel of $2.56 per foot. Items included in cost of shovel operation are depreciation, interest, power, repairs, and supplies. The shovel speeds up drifting considerably, especially after chute raises are started on the level, which is done as soon as each heading has advanced about 100 ft.

Chute or grizzly raises are spaced 40 ft. apart along the level, and, until they have been advanced high enough to install the chutes, all broken ore from the raises falls to the level, and must be shoveled quickly or the work of driving the headings will be impeded. The shovel is used generally to muck up the broken ore from raises on the shift when level headings are being drilled, so that the track will be clear for shoveling headings after blasting.

Development work in rock consists of shafts, shaft stations, ore and rock pockets, crusher stations, pump stations and sumps, underground shops, bypass drifts, and crosscuts to oreshoot from shaft stations. Development work in ore consists of haulage or tramming levels only. Such work as tramming-level chute raises, pillar raises, pillar drifts and crosscuts, sublevels and sublevel crosscuts, grizzly chambers, cone or draw raises, mining raises, and stope undercutting are charged to an account called "stope preparation," a subdivision of the development account. However, all stope preparation is in ore. Table I shows amount of rock excavated in development and Table II shows amount of ore excavated in development and stope preparation per ton of ore developed and prepared for stoping on each level, for a level interval of 300 ft.

Table I—Development in Rock

	Linear Feet of Work	Tons Rock Broken
Main hoisting shaft	300	6,528
Ventilation shaft	300	3,264
Main shaft station	50	1,912
Pump station and sump	150	2,000
Rock and ore pockets	150	2,000
Underground shop	75	2,300
Crosscut to oreshoot	50	575
Bypass tracks at shaft station	200	1,466
Crusher station	75	2,000
Totals	1,350	22,045

Table II—Development and Stope Preparation in Ore

	Linear Feet of Work	Tons Ore Broken
Haulage or tramming level	2,500	22,500
Sublevel, crosscuts, and grizzly chambers	3,632	12,448
Pillar raise, crosscuts, and drifts	3,776	12,944
Chute or grizzly raises	1,120	4,264
Cone or draw raises	1,120	47,192
Mining raises	3,552	12,416
Stope undercut	2,148	35,728
Totals	17,848	147,492

The length of tramming levels is 2,500 ft., the thickness of the oreshoot is 25 ft., and with a level interval of 300 ft., measured on the dip, the amount of ore in the block or between levels is 1,785,700 gross tons. As about 85 per cent of this ore will be recovered, the net ore to be recovered from each level is about 1,517,945 gross tons.

The present charge covering the cost of development in rock per ton of recoverable ore is 20c. and that for the cost of development and stope preparation in ore per ton of recoverable ore is also 20c., a total of 40c. per ton. The complete mining of one level is expected to prove that this charge can be reduced to 30c. or less. Linear feet of work and tons of rock and ore broken in development and stope preparation per level, or per 1,517,945 tons of recoverable ore, are shown in Tables I and II.

Table I shows that 22,045 tons of rock is broken in development. Inasmuch as the tonnage of recoverable ore between levels, where the level interval is 300 ft., is 1,517,945, the tonnage of rock broken in development per ton of ore recovered is 0.0145. Table II shows that 147,492 tons of ore is broken in development and stope preparation. Ore broken in development and stope preparation per ton of recoverable ore is 0.097 ton, or 9.7 per cent of the total amount of ore. Deducting the 147,492 tons of development and stope preparation ore from the 1,517,945 tons of recoverable ore leaves 1,370,453 tons of ore to be stoped or recovered from pillars.

From stopes 1,072,000 tons of ore is mined and from pillars 298,353 tons of ore is recovered. The percentage of ore recovered from all sources and where the level interval is 300 ft., is as follows: Development and stope preparation, 9.7 per cent; stoping, 70.6 per cent; and pillar recovery, 19.7 per cent.

(To be continued)

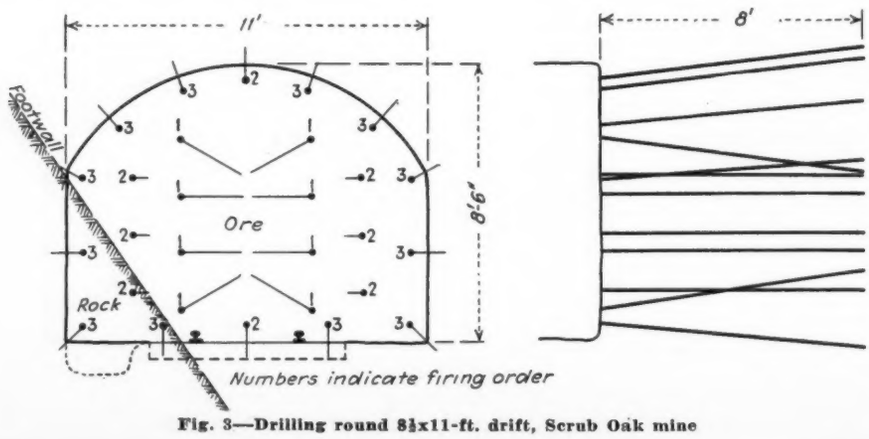

Fig. 3—Drilling round 8½x11-ft. drift, Scrub Oak mine

Geophysics From the European Viewpoint

A Review by Sherwin F. Kelly

TRAITÉ PRATIQUE DE PROSPECTION GÉOPHYSIQUE (Practical Treatise on Geophysical Prospecting). *By C. L. Alexanian. Paris, France: Librairie Polytechnique Ch. Béranger; pp. 268; illus.*

ÉTUDES GÉOLOGIQUES ET PROSPECTIONS MINIÈRES PAR LES MÉTHODES GÉOPHYSIQUES (Geological and Mining Studies by Geophysical Methods). *By P. Geoffroy and P. Charrin. Algiers, Algeria: Geological Survey of Algeria; 4th Series, Geophysics, No. 1; pp. 346; illus.*

The purpose of these two books is to give to a geologist or an engineer a simple account of the science of geophysics. A large proportion of Alexanian's book is devoted to theoretical considerations, whereas Geoffroy and Charrin emphasize actual field examples. This is, perhaps, the best indication of the different attitudes of the authors toward their subject. "Traité Pratique" is divided into two parts. In the first the author presents the gravimetric, magnetic, electric, seismic, geothermal, and radioactive methods of prospecting, discussing the theory upon which each is based and the instruments employed. Mathematical treatment is not avoided, and pertinent formulas are developed or given. His information is not always up to date, however, as when he describes only the microphone type of receiver as being the accelorometer in use in seismic prospecting, whereas this type is actually being abandoned. Nor does he show much familiarity with recent developments in electrical prospecting, most of his material being drawn from old publications. When taking up examples of practical application of these geophysical methods, the same division is followed, each method being dealt with in turn, so that it is the method rather than the field problem which is emphasized. The American reader will recognize, among the examples cited, data already published in the United States, but he will also find some interesting European surveys described. This part of the book is unevenly balanced, several field examples being given for some methods, whereas to illustrate the seismic method the author describes only two small surveys by the reflection method! He then states that the reflection method has been practically abandoned in favor of refraction shooting, which happens to be the exact opposite of what is occurring in the extensive seismic work on the American continent. The author describes, in a general way, the exploration work in the Gulf Coast area. He is equally unfortunate in his statements about torsion-balance work when describing the experimental survey made near Picher, in the Tri-State district. The assertion is made that the gravity highs were "due to the presence of ores and to the calcareous formations which generally accompany them in the region prospected." As a matter of fact they occur in the silicified beds, which give gravity *lows*, and drilling failed to find any ore at all in the particular case cited. A serious typographical error is to be noted on p. 207, where Alexanian says the "E Body" and "C Body" of the Noranda Mines were discovered by spontaneous polarization. It should be "G Body," as on the accompanying map, not "C." He also confuses the "sondage électrique," or "electrical drilling," and the "carottage électrique," or "electrical coring," two entirely different processes for quite different purposes. Bibliographies are placed at the end of each chapter, an excellent idea, but the author's citations are not always well chosen. The Alexanian book would be useful to a student desiring a theoretical introduction to the various methods of geophysical prospecting, provided he keeps in mind the shortcomings of the practical section.

The case is otherwise with the Geoffroy and Charrin book. They devote the first part of their work to a brief consideration of the various prospecting methods, the space consecrated to this aspect of the subject being a smaller proportion of the book than in the one just reviewed in the foregoing. Mathematical expositions are reduced to a minimum, and little space is given over to a discussion of apparatus. Mathematics and apparatus are briefly taken up in appendices, for those desiring greater detail of exposition. Enough is given, however, to make clear what follows in the second part of the volume, where the practical application of geophysics is considered. The attack on this subject differs radically from Alexanian's. The interest of the authors here is not in how a given method of prospecting works in the field, but in how a given geological problem was solved. Consequently, in a chapter entitled "Study of Horizontal Formations," the use of electrical and seismic methods is described. Similarly, grouped in the chapter on "Tectonic Studies," are examples of gravimetric, electric, magnetic and seismic surveys, and how they have been used, individually or in concert, to solve the problems posed. The same plan is followed in discussing the investigations of salt domes, the prospecting for metallic ores, and the observations which can be made in drill holes. In connection with the latter, space is given to a description of the so-called "electrical coring," which is proving so useful in oil work. A chapter is also devoted to various detailed studies, such as of the different beds of a syncline, of faults, of clay beds in sand. A well-chosen bibliography is at the end of the book. The authors make a valiant effort to get L. Spraragen's name correctly spelled. In the text he is Sparangen; in the errata and the bibliography he is Sparagen; but nowhere is he Spraragen! Also, one is led to wonder why, in almost all bibliographies on the resistivity method, the early and fundamental work of Wenner is consistently ignored! Would the reputation of later workers be dimmed by citing it?

The Noranda comes in for confusion again, the authors in this case giving more credit than is due the electrical method when they state, p. 254, that the "E, H and G ore body" were not found until after the electrical exploration. If the reviewer, who performed that survey, remembers correctly, the "H" body was being drilled *while* he was doing the electrical field work. Some skepticism is expressed by the authors regarding the statements being made as to the reduction in the size of dynamite loads now required in seismic shooting. They would probably be astounded at the minute charges being used in this country. One is also inclined to wonder if the writers of this book were skeptical as to the validity of electrical surveys carried out by anybody but the Schlumberger organization, as practically all of the electrical explorations they cite were Schlumberger ones, with a few of Lundberg-Sundberg origin, others being ignored.

To the American reader, a few examples of field work described were familiar, but many of them, being entirely of European origin, have the refreshing air of novelty. The Geoffroy and Charrin book can be recommended to geophysicists and geologists who want to obtain a good picture of what the science and art of geophysics can do for geology.

GEOLOGY OF THE COUNTRY SURROUNDING VRYHEID. *By W. A. Humphrey and L. J. Krige. Pretoria, South Africa: Geological Survey. Price 5s.*

This bulletin contains an explanation of the geological map of an area of about 2,100 square miles around Vryheid, Natal, much of which is covered by rocks of the Karroo system. Gold has been found there. Manganese occurs as a surface enrichment of a manganiferous shale. Impure graphite is known. Saltpeter is found in the form of sheets and crusts in and on sandstone. Coal constitutes the principal economic mineral.

GRUBSTAKE. *By Mark L. Requa. New York: Charles Scribner's Sons. Pp. 360, $2.*

A prominent member of the mining profession draws on extensive experience to produce a fictional work of absorbing interest. Besides being factual as to place names and historical associations, the volume carries an exciting tale, in which the romance of the early mining days in Nevada lives again. It would seem that Mr. Requa, after the toil of the day, draws up his chair before the hearth of the open fire, and, gazing at the glowing embers, sees fact and fancy. These pictures he has captured and woven together for the delight of others. And the guess is hazarded that he too derived much of pleasure from "Grubstake."

Bearing as it does the ring of authenticity as to locale and incident, the book will revive the memories of prospectors and mining engineers who participated in that stirring chapter. Some readers, however, will regret that Mr. Requa did not chronicle actual dates and events, instead of fictionizing. However, the book will have particular interest at this time when so much attention is being paid to gold hunting. Mr. Requa's story contains much technical data which will be a revelation to the average reader and of real value to such as are harking to the call which in '74 proved strong enough to make men dare the dangers this book so graphically describes.

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EDWARD ORTON, JR.—A Memorial. *Columbus, Ohio: Engineering Experiment Station, Ohio State University.*

This memorial to the former dean of the College of Engineering of Ohio State University, who died in February, 1932, has been prepared by a faculty committee. Son of Edward Orton, educator and geologist, Edward Orton followed in his father's steps in each capacity and succeeded him as State Geologist of Ohio in 1899, a position held until 1906. Studies of the nonmetallic resources of the State marked the years of his connection with the State Survey. He had a large part in the expansion of the clay-working industries in Ohio and made notable contributions to ceramic literature. His many achievements included those of causing the University to create a department of ceramics; of initiating the move which led to the setting up of the engineering experiment station at Columbus; and of establishing the Edward Orton Memorial Library of Geology, in his father's memory. His War record also was illustrious. He was one of the four who helped prepare the National Defense Act and urged its passage. The volume contains tributes to his career and appreciations from several sources.

SOUTH AFRICAN MINING AND ENGINEERING YEAR BOOK. *Edited by S. R. Potter. Johannesburg: S. A. Mining Journal Syndicate; 10s. 6d., postage 2s. extra. London: Argus South African Newspapers, Ltd.; 12s. 6d. post free.*

RHODESIAN MINING AND ENGINEERING YEAR BOOK. *Edited by S. R. Potter. Johannesburg: S. A. Mining Journal Syndicate; 10s. 6d., postage 2s. extra. London: Argus South African Newspapers, Ltd.; 12s. 6d. post free.*

These valuable works of reference are issued at yearly intervals, and the present volumes indicate that standards are being maintained. Each is supplemented by maps of the Witwatersrand gold fields, of the coal and iron mines and works of the Union, and of the power facilities of the Union and of Rhodesia. Text matter contains details of all the more important mines and financial companies in British South Africa. In the Directory Section may be found a list of consulting engineers and geologists, mine buyers, and manufacturers, and much information of general interest.

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DIRECTIONS FOR THE USE OF THE ASKANIA TORSION BALANCE. *By C. A. Heiland. Texas: American Askania Corporation, Houston. Pp. 88; illus.; $3.30.*

Dr. Heiland and his co-authors, which include J. A. Malkovsky, Dart Wantland, and John H. Wilson, have not set out to write an elementary introduction to the art of torsion-balance surveying. The reader who undertakes to study this book must be prepared for a mathematical treatment of the subject, although the authors say that they have handled it so that knowledge of potential theory is not necessary. A brief résumé of torsion-balance theory serves as an introduction to the book; from there on the work is meant to aid the field operator in his manipulation of the balance. It is more than a handbook of mechanical directions, however, as may be judged from the chapter titles: "How to Compute Gravity Data From Instrumental Records," in which are included directions for reading the plates, and the working equations for various positions; "How to Compute Corrections"; "How to Plot the Corrected Results"; "How to Manipulate the Instrument"; "How to Do Torsion Balance Field Work"; "Interpretation of Results"; "Cost of Torsion Balance Work," which deals with the average cost in the Texas Gulf Coast region.

In addition to its usefulness as a field manual, this book can be highly recommended to any serious student of geophysics who desires to get a good systematic picture of the whole process of torsion-balance surveying. It is from this point of view that the reviewer is led to express a hope that the Askania company will follow the present work with other and similar presentations on their magnetic and seismic instruments.

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ELEMENTS OF MINING. *By Prof. Robert S. Lewis. New York, N. Y.: John Wiley & Sons, Inc., pp. 510; illus.; $5. London: Chapman & Hall, Ltd.*

To cover so complex a subject as mining in a single volume of moderate size calls for brevity. Descriptive text must be clear, moreover, especially in discussing mining methods, because here the student must think in three dimensions, something not always easy. Professor Lewis has turned out a book noteworthy for its terseness and clarity of expression. The omission of details not immediately required in an elementary course is balanced by copious references to mining literature, for the most part fairly recent. Perhaps in no other period of equal length has such a wealth of data appeared on mining practice as in the last five years. Each of the fifteen chapters concludes with an extensive bibliography. The author's treatment of his subject is indicated by the contents table: Prospecting; Mining Law; Subsidence and Support of Excavation; Explosives; Drilling and Blasting; Shafts; Hoisting; Mining Methods; Mine Haulage; Mine Drainage and Pumping; Compressed Air; Ventilation; Sampling and Valuing Mines; and, finally, Organization, Management, Safety Work. Underground mining methods he classifies under the headings "open stopes," "supported stopes," and "caving methods," shrinkage stopes being included under the first on the ground that they are "primarily unsupported." Combined methods are not considered, in order to simplify the treatment. The book is amply illustrated, sketches and diagrams predominating.

The choice of title, "Elements of Mining," at first seems a bit unfortunate as inviting confusion with the widely known book, of the same title, by George J. Young. Ample precedent for such duplication of title doubtless is to be found, however, in many lines of literature. A. H. HUBBELL.

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ELECTROMETALLURGICAL RESOURCES OF NORTH PLATE RIVER BASIN, WYOMING. *By C. S. Dietz. Cheyenne, Wyo.: Wyoming Geological Survey. Price $1 postpaid.*

This treatise is not offered as a geological or technical description, but is designed to provide the student, the consumer, the investor, and the producer with a compilation of the mineral resources most feasible for hydroelectric development in southeastern Wyoming.

Personal Notes . . . Comment . . . Criticism

George **A. Easley,** now in Europe, is expected to return to New York shortly.

H. E. Nichols, of London, is reporting on the Nullagine banket deposits in North-Western Australia.

Mark L. Requa's fictional work "Grubstake," which is reviewed elsewhere in this issue, has become a best-seller on the Pacific Coast and is in its second printing. The book includes a friendly dedication to **Herbert Hoover.**

S. J. Truscott, of the faculty of the Royal School of Mines, a past president of the Institution of Mining & Metallurgy, and well known throughout the world for his treatises on ore dressing, and on mining in South Africa, is visiting several mining regions in the United States.

W. Rex Storms, who concludes in this issue a series of articles on diamond-drilling practice, a graduate of the College of Mines of the University of Washington, is geologist to United Verde Copper. The three articles in the series have been abstracted from a thesis submitted to the University of Washington, on the acceptance of which he was granted the degree of Engineer of Mines.

A. J. Keast, manager of Beattie Gold, reports that the initial 800-ton unit of the mill, the largest initial unit to be built in Canada to date, is being tuned up, and the mine being prepared for the required output. The extent of the ore deposit warrants a mill several times the size of the one soon to go into operation, and further units will be added after experience has been gained with the first.

Huntington Adams, who in this issue proposes an international gold unit, was educated at Harvard, and served a preliminary apprenticeship in various subordinate positions at mines in the United States and Canada prior to appointment as general manager of operations for Cia. Minera de Natividad y Anexas, Mexico, 1908-12. From 1913 to 1922 he intermittently practiced as a consultant in New York and directed nitrate developments in Chile for various interests, serving during the War as consulting engineer to the Ordnance Department, United States Army. In 1925 he was appointed chief consulting engineer for Patiño Mines & Enterprises, Bolivia, subsequently acting as special delegate in Bolivia for United States bankers. He is an alternate director of Roan Antelope, Mufulira Copper, and other Rhodesian companies; and is expected to return to New York from Europe early in May.

Charles A. Rose, who describes in this issue the occurrence and treatment of a high-acid gold ore in Cyprus, was graduated from the University of Nebraska in 1902. Professional work followed with Homestake Mining and Alder Creek Gold Mining. In 1903 he was appointed chief chemist for Boston & Colorado Smelting; in 1909, research chemist for the Colorado division of the A.S.&R. In 1911 he took charge of the experimental laboratory at Perth Amboy for the development of a process for the treatment of the Chuquicamata ores of Chile Exploration, and in 1913 was appointed metallurgical engineer for Chile Exploration and Braden Copper, a position held until 1916. From 1916 to 1923 he acted as assistant consulting metallurgical engineer to Guggenheim Brothers, then general manager for British-America Nickel Corporation (1923-24), and manager of the research department of A.S.&R. (1924-31). He recently returned to New York from Cyprus, where he conducted metallurgical research for Cyprus Mines Corporation.

A. F. Brigham has spent a large part of his time in developing the Young-Davison property at Matachewan since he relinquished management of Hollinger Consolidated and became the company's general consultant. The Matachewan field has been a puzzle to Canadian gold miners for fifteen years, and Mr. Brigham is taking steps to solve this by using a pilot mill, now being erected, to sample in bulk the large gold deposits of the region. This bold step seems likely to prove well justified. Favorable results have followed a similar course of action on other properties in Ontario and Quebec.

Sherwin F. Kelly, who contributes to this issue a review of two recent treatises on geophysical prospecting, received his education at the universities of Paris, Kansas, and Toronto, with subsequent teaching experience on the faculties of the two latter institutions. He was a pioneer in the introduction of electrical prospecting in the American continent, and since 1930 he has practiced as a consulting geologist and geophysicist, with headquarters in New York.

L. M. Banks has joined the staff of Shenandoah-Dives Mining, operating at Silverton, Colo.

Alan M. Bateman has returned to New Haven after a professional trip to Arizona and New Mexico.

T. H. Bayldon recently took up his duties as general manager of Tanganyika Central Gold Mines, Selenke, Territory of Tanganyika.

T. Skewes Saunders has changed office headquarters in Mexico City from Avenida Cinco de Mayo 10 to Venustiano Carranza 37, Despacho No. 4.

R. J. Morgan, Ralph Symons, A. G. E. Turnly, and **O. B. Soskice** left London on March 10 for the properties of Chosen Corporation, at Talyudong, Korea.

Ralph Hayden, superintendent for many years of the Quincy mill, at Mason, Mich., has resigned and gone to Boston, Mass., to engage in consulting engineering work.

Charles K. McArthur has been appointed manager of the Toronto office of The Dorr Company, vice **Henry W. Shoemaker,** who recently resigned to return to private engineering practice.

Clyde A. Heller, president of Tonopah Belmont, recently inspected the Vallecitos property at Murphy, Calif., being developed for the company by **Frederick Bradshaw,** of San Francisco.

Sir Herbert Gepp has been appointed technical consultant to the Mount Morgan company, Queensland. **T. M. Owen,** consulting metallurgist, Wiluna Gold Mines, has been named a director.

Prof. Corbin T. Eddy, of the metallurgical faculty of the Michigan College of Mines & Technology, recently returned to Houghton, Mich., after a visit to England, France, and Germany.

H. Houghton Lauer, formerly chief engineer with the Atlas Portland Cement and Universal Cement companies, is now in practice as a consultant, with headquarters at 41 East 42d St., New York.

R. E. Crockett, who collaborated with H. M. Roche in the authorship of the article on "Iron Ore Mining and Milling at Scrub Oak" in this issue, the first of a series, was graduated from the Cooper Institute, became assistant manager for Ringwood Mining, 1919-29, and has since been general superintendent of the Dover Division of Alan Wood Mining.

H. G. Price, manager of Kuala Lumpur Tin Dredging, F.M.S., is in Australia.

G. Gordon Thomas is manager of Elandshootge Proprietary, Transvaal, South Africa.

N. F. G. Davis has returned to Bechuanaland, after a visit to Vancouver, B. C.

Rudolph Agnew, of London, is inspecting properties at Wiluna and Laverton, Western Australia.

E. W. Beatty, president of Canadian Pacific Railway, has retired from the board of International Nickel.

Oscar A. Hershey has returned to San Francisco from a professional visit to the gold-mining districts of Ontario.

G. B. Hogenraad is in Java, where he expects to remain for several months before returning to Indaroeng, Sumatra.

G. W. Eaton Turner, general mines manager for Ashanti Goldfields, Gold Coast Colony, is returning to London in April.

M. A. Newman recently examined mining properties in Nevada and Markleville, Calif., for Pacific Coast clients.

W. H. Trewartha-James has returned to England from Western Australia after a study of metallurgical problems at the Great Boulder Proprietary.

Charles Bocking, president and general manager of Granby Consolidated Mining & Smelting, was recently elected president of the Mining Association of British Columbia.

R. J. Parker, assistant to the managing director of Roan Antelope and Rhodesian Selection, is in the Kenya region. He will return to London after visiting the Roan Antelope property in Northern Rhodesia.

J. S. Godard, who resigned last year from the Department of Mines at Ottawa to become metallurgist with Canadian Industries, has joined the metallurgical staff of Lake Shore, at Kirkland Lake, Ont.

H. M. Roche, one of the authors of a series of articles, the first of which appears in this issue, on iron-ore mining and milling in the Highlands of New Jersey, gained a broad experience in California, Nevada, Mexico and British Columbia before appointment in 1912 as superintendent of the Mount Hope mine, in New Jersey, where he initiated important changes in underground practice that attracted wide attention. In 1917 he became general superintendent of mines and quarries of Replogle Steel, and developed the Scrub Oak mine for production of bessemer ore and designed the original mill. He has since specialized in mill design and construction, and has contributed, by the invention of magnetic-separation apparatus, to the art of concentrating iron ore. His professional status in this specialty may be gaged from the fact that of the twelve modern magnetic iron-ore mills in the United States he designed, constructed, and has operated five.

Paul M. Sorensen, formerly metallurgist and mill superintendent for Hirst-Chichagof Mining, was recently appointed manager of the Alaskan properties of that company, succeeding the late **D. J. Williams.**

Walter E. Burlingame, formerly of E. E. Burlingame & Company, and more recently general manager of the Smuggler mine, in Colorado, has opened an engineering office and laboratory at 2040 Broadway, Denver, Colo.

Dr. Sidney W. Smith, assayer to the Royal Mint, president of the Institution of Mining & Metallurgy, recently arrived in Canada from London to attend the annual meeting of the Canadian Institute of Mining and Metallurgy at Toronto.

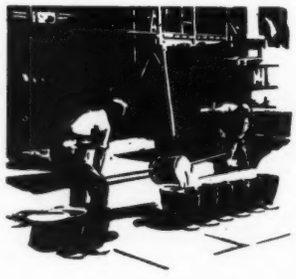
John A. Agnew, a director or chairman of the board of mining companies operating in Africa, Mexico, and Australia, has been appointed chairman of the boards of Consolidated Gold Fields of South Africa and of New Consolidated Gold Fields of South Africa.

Paul A. Sexson, who for the past two and a half years has been secretary to P. G. Beckett, vice-president and general manager of Phelps Dodge Corporation, has left Douglas for Palo Alto, Calif., where he will become a member of the secretarial staff of ex-President Hoover.

Floyd H. Viets returned recently to New York following an engagement of four years with Cerro de Pasco Copper, in Peru, during which period he supervised the design and construction of smoke abatement works. Mr. Viets, after a short visit in New York City, left for Los Angeles, Calif., to resume consulting practice.

Henry S. Beal (Dartmouth College and the University of Berlin) has been elected president of the Sullivan Machinery Company, vice **Arthur E. Blackwood,** who becomes chairman of the board. Mr. Beal entered the employ of the Jones & Lamson Machine Company as machinist in 1909, later becoming general manager, a position from which he recently resigned to join the Sullivan organization.

Dr. W. C. Mendenhall, United States Geological Survey, Washington, D. C., reports that the following topics will be discussed at the forthcoming session of the International Geological Congress to be held in Washington, July 22-29: Measurement of geologic time; batholiths and related intrusives; zonal relations of metalliferous deposits; major divisions of the Paleozoic era; geomorphogenic processes in arid regions; fossil man and contemporary faunas; orogenesis; geology of petroleum; copper resources of the world. Membership in the congress is open to all interested persons, and full details of the meetings and excursions may be obtained from Doctor Mendenhall.



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Obituary

Alexander Montgomery, State mining engineer, Western Australia, 1902-29; at Perth, W. A., on Feb. 15; aged 71.

Daniel C. Babbitt; in Peru, on Feb. 7; aged 65. Mr. Babbitt was controller of Cerro de Pasco Copper, and a graduate of Lafayette College, Easton, Pa., class of '91.

James M. Holman, chairman and managing director, Holman Brothers, prominent in the development of the Cornish mining industry, sometime governor of the Camborne School of Mines; at Camborne, England, on Feb. 27.

H. T. Curran (South Dakota School of Mines, 1906), formerly with Mainstay Gold Mining & Milling, in South Dakota; Guanajuato Amalgamated Mining & Milling, Mexico; and later in practice at Los Angeles; at Great Falls, Mont., Feb. 16.

J. L. Phillips, formerly superintendent of the Gold Roads mine, Kingman, Ariz., associated with companies operating in Nicaragua, Mexico, and the South West, member of the first graduating class of the Missouri School of Mines and Metallurgy; at Los Angeles on March 3.

George H. Birch, formerly president of Dan Creek Mining; at New York on March 23, after an operation; aged 51. Mr. Birch and his brother, **Stephen Birch,** pioneered in the development of mining properties in the interior of Alaska. He was an alumnus of the Columbia School of Mines.

James H. Jowett, director and executive vice-president of the Ingersoll-Rand Company; on Feb. 16, in New York; aged 58 years. Mr. Jowett was educated in the public schools of Cleveland, and was first employed by the Ingersoll-Sergeant Drill Company, now Ingersoll-Rand, with which he has since been associated.

J. Leslie Urquhart, chairman of Russo-Asiatic Consolidated Mining and Mount Isa Mines, director of New Guinea Goldfields; in London, of pneumonia, on March 13; aged 58. Mr. Urquhart, who received his mining education at Glasgow and Edinburgh universities, practiced his profession at first in the oil fields of eastern Europe, then becoming the head of vast mining interests in Russia, sequestered by the Soviet government after the revolution of 1917. After futile attempts to secure their return or compensation, he became head of the Anglo-French company, operating in Turkey. Latterly his interests have included the Mount Isa property in Queensland, New Guinea Goldfields, and the Northern Rhodesian copper fields.

Gold Currency Unit for International Exchange

The Editor:

A disturbing feature in international credit is the insecurity in the gold value of all currencies. Only a few nations are still on an effective gold standard, redeeming their currencies in gold at par without restrictions on exchange movements or shipments of gold; only in those countries can gold values be deposited subject to withdrawal abroad when required. Some nations maintain their exchange at theoretical gold parity, but prohibit or restrict exchange and withdrawal of gold; this amounts only to maintaining gold prices, for the gold standard implies redemption in gold. Even the maintenance of gold prices is not successful everywhere; a bootlegging of exchange at rates below par is practiced in some countries. Others are frankly off the gold standard in their exchange, and the gold value of their currencies remains, therefore, governed only by their trade balances; restriction of foreign trade and financing becomes essential under these conditions if wide fluctuations in exchange rate are to be avoided.

The result has been a concentration of liquid capital, from all over the world, in the central banks of the countries still on a gold basis, and a consequent reduction of gold reserves in the countries where it is most needed. In so far as the quantum theory applies to gold—and undoubtedly it does to a considerable extent—the withdrawal of gold from non-gold-standard countries has tended to lower world prices in gold—to raise the value of gold, because of its scarcity, in terms of commodities—and thereby to accentuate the depression in gold-standard countries by lowering the world prices of those of their products dependent on world prices, such as agricultural products in the United States.

The support of currency has indicated recognition of relative degrees of insecurity, rather than confidence, even in gold-standard countries. Since Britain was forced to abandon the gold standard in September, 1931, followed by Japan and other nations, it has been apparent that any of the remaining gold-standard countries might follow the same course, either by being forced to it unwillingly as a result of internal developments, such as overborrowing, or as a result of deliberate intention in an attempt to mitigate the effects of the crisis by thus artificially increasing internal prices.

By a sudden unilateral decision of the British Government to abandon gold, foreign depositors lost 25 per cent of the gold value of their deposits almost overnight, and later 30 per cent. The Bank of France alone is estimated to have lost thereby the equivalent of 80 million dollars. As a result, foreign depositors have become nervous; after the establishment of the Reconstruction Finance Corporation in the United States during the first half of 1932, European central banks converted their American balances into gold and withdrew them as rapidly as was feasible.

Excess liquid capital from countries outside the gold standard is probably deposited mostly in Europe; but, should a crisis be threatened, perhaps as a result of political developments, a new flight of gold in another direction may result. The situation indicates a lack of confidence in currency values throughout the world. Any step that will help to restore confidence will increase the possibility of improvement in world conditions.

Under the gold standard, as it has been employed, commitments have been expressed in currency units. In international transactions these have been usually in the currency units of the creditor country. This custom, which has come down from earlier times, when ideas of banking were more provincial than they are now, seems rooted in the supposition that transactions made in this form extend the economic influence of the creditor country, and perhaps make securities more marketable among that country's investing public. The futility of the former idea is illustrated in Britain's abandonment of the gold standard. As for the latter, it can hardly be important in these days when securities are quoted, and sometimes issued, in all sorts of uneven prices.

Under the gold standard, currency units themselves are based on specified weights of pure gold, or, what amounts to the same thing, on specified weights of gold of a specified fineness. The specification of the weight of gold in a currency unit, however, is the result of legislation by the individual nation. Therefore, it is subject to alteration by subsequent legislation. The weights of gold equivalent to the currency units, on the other hand, are physical units of world-wide acceptance, not subject to alteration by unilateral legislative action.

If, therefore, financial commitments should be expressed in weight units of gold instead of in currency units, which in turn are based on the same weight units, the debts would be free of danger of being altered by any one government. Short of repudiation, the gold value of debts could not be altered.

The expression of financial commitments in weight units of gold would have two important advantages: (1) It would eliminate the danger of alteration by unilateral legislative action, and would inspire confidence thereby; (2) it would provide an international unit of gold value available for international transactions by countries now off the gold standard, to which it would prove an advantage; for however much an internal currency may vary in gold value, foreign trade is still conducted on the basis of world prices, which have no other base in the modern world except gold. This fact is well exemplified in the case of China, which is on a silver basis of internal currency, and which, because of the drop in the price of silver, has had its internal money value greatly reduced in terms of gold. Nevertheless, its whole foreign trade is conducted on a basis of gold prices. Nor is the idea of a dual currency, with one system for internal use and another for external, a new one. It has been in successful use in Argentina for many years, where a "gold peso" (now off parity) is used for external transactions, and a "paper peso" *(moneda nacional)* for internal operations.

The use of an international gold unit would enable a center of financial and trade exchange such as London to conduct its business in money that would provide security in gold value to its foreign commitments, without affecting Britain's freedom to regulate her internal currency value to suit herself. In this way it should go a long way to meet the desire of the American Government that Britain return to the gold standard, without the dangers which the British see in an outright gold stabilization of the pound sterling under present conditions.

The unit of weight of gold to be used for an international unit is not of itself of primary importance; but the adoption of a convenient and internationally acceptable unit would facilitate general adoption. The troy ounce, an antique French measure, is used now only by the English-speaking peoples. Furthermore, that weight of gold is too large to make it a satisfactory unit of value. The pennyweight, another troy measure, is used only by the British. The gram of the metric system, now the official unit of weight in all countries of the world except the English-speaking ones, extensively used among scientists everywhere, would make a convenient unit of value. Two currencies only are based directly on it—the Mexican peso and the Japanese yen, both of which at par are the equivalent of $\frac{3}{4}$ gram of pure gold. Both of these nations, however, are now off the gold standard.

A gram of pure gold is worth exactly $0.6646 in United States currency, or quite closely $\frac{2}{3}$ of a dollar (within 3 thousandths). It is worth closely $\frac{1}{3}$ of £1 sterling at present gold exchange (£0.13657, or about 7$\frac{1}{3}$ to £1 at the old parity). Similarly, it is worth 16.96 French francs, 2.69 Reichsmarks, or 3.444 Swiss francs.

Objection may be raised that such a unit would not have an even exchange value into any currency. This, however, seems to be one of its advantages, because thereby it would be equally fair to all concerned. Furthermore, no existing currency has an even exchange value into any other, and no securities ever have an even value after they have been issued and subjected to the valuation of market operations. Obviously, gold-gram securities could be quoted in each country in terms of its national currency so far as this might be found advantageous for market use, as are foreign currencies.

The first advantage of the gold gram would be international security in gold value. The second would be the establishment of a gold money unit of world-wide acceptance, independent of any nation. It might well become the world money unit of the future.

Menton, France. HUNTINGTON ADAMS.

INDUSTRIAL PROGRESS

Hydraulic Speed Control Features Drive of Broken Hill Hoist

Use of an hydraulic coupling in a motor-generator set that supplies direct current for driving an electric hoist installed at a shaft of North Broken Hill, Ltd., is a detail of hoisting practice that will be new to many. This application is described by F. J. Mars in an article on the central power plant at Broken Hill, published in the Proceedings (Sept. 30, 1932) of the Australasian Institute of Mining & Metallurgy. The hoist is equipped with Ward Leonard control, the drums being driven by a direct-current motor which receives its power from the motor-generator set of which the coupling is an element. The converter set consists of four components in one line in the following order: A synchronous-induction motor, the hydraulic coupling, a load-equalizing flywheel, and the d.c. generator.

The coupling is the central element of the so-called Vulcan-Sinclair hydraulic speed control. It is made by the Hydraulic Coupling & Engineering Company, Ltd., Isleworth, England. In the Broken Hill case it acts as an hydraulic slip regulator. It is constructed as follows, according to the maker:

Its characteristic is a scoop-tube and a circulation system by which infinitely variable speed regulation is attained. The driving half or impeller (1) is mounted on the driving shaft. Opposite to it is the driven half or runner (2) carried on the driven shaft. The working liquid is retained by the inner casing (3), and an outer casing (4) is provided to form a chamber in which the scoop-tube (5) is housed. Elements (1), (3), and (4) are bolted together and rotate with the driving shaft as one unit. Runner (2) rotates with the driven shaft. The scoop-tube is stationary, being supported from a bracket. No mechanical connection exists between impeller and runner halves, and the power is transmitted by the kinetic energy of liquid discharged by the impeller directly against the vanes of the runner. It naturally results that the speed of the runner lags slightly behind the impeller, and a vortex ring of circulating liquid constitutes the driving connection between the two halves. When full of liquid, the "slip," or lag, is of the order of 2 per cent, the driven shaft then being at its maximum speed. By running partly full of liquid, the speed of the driven shaft may be decreased as desired; and by allowing the coupling to empty, the driven shaft may be disconnected from the driving motor. Hence the latter can be started up light and the load applied by admitting liquid. The coupling can be adjusted to operate up to about 15 per cent slip.

The method of starting the hoist at Broken Hill, according to Mr. Mars, is as follows: The synchronous motor (selected because it permits control of power factor and the use of high voltage, in this case 6,900, without transformers) is started and brought up to speed as an induction motor, then excited for synchronous operation with the coupling empty—that is, without transmission of power to the flywheel and the generator. The latter two components are next brought up to speed by admitting oil to the coupling. This oil passes through a valve controlled by a small torque motor whose field is energized by series transformers in the line circuit of the synchronous motor. The degree of filling of the coupling is therefore a function of the load on the synchronous motor, and the load can be equalized to any predetermined maximum value. Power input from the transmission line is therefore limited, the energy for the hoisting peaks being supplied by the flywheel.

This hydraulic speed control is applicable also to boiler house fans, feed pumps, and industrial drives in general.

■

Gold Gravel Washing Devices Multiply

Portable plants, such as shown in one of the accompanying illustrations, for washing gold-bearing gravels are being made by the Diamond Iron Works, Minneapolis, Minn. One of these plants consists of a hopper and grizzly, a mechanical feeder delivering to an inclined belt conveyor which in turn feeds a revolving screen, the oversize of which is sent to waste via a tailing conveyor or stacker and the undersize of which goes to four, six, or eight (depending on the size of the unit) centrifugal bowls for concentration, the entire unit being mounted on wheels which run on a temporarily laid track. Gravel is delivered to the hopper by a shovel or a dragline, which also serves to push the washing unit along the track to advance it. The grizzly bars are spaced 8 to 10 in. The waste conveyor is 50 ft. long and has a 24-in. belt. It can be swung 90 deg. right or left. The centrifugal bowls have circular riffles on the inside. The sand overflowing the bowls is dewatered on a drag washer and delivered to the tailing conveyor. Except for the waste conveyor, the plant is operated by a 40-hp. gasoline engine. The conveyor is run by a separate 15-hp. unit. A four-bowl plant is said to be able to handle 800 to 1,000 tons in 10 hours.

Several of these units are said to be working in the Black Hills of South Dakota. At Custer, the Eureka Mining Company is working three units on 30c. gravel and securing, according to report, about 20 oz. gold per day. Five men are required to operate one of these units.

Portable plant for washing gold gravels, as installed on property of Grand Hill Mining Company, Custer, S. D. Its essential features are a revolving screen and centrifugal riffled bowls

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New Portable Dry Washer

A portable dry washer, weighing only 75 lb., has been developed by W. H. Harris, of Imlay, Nev., who is shown operating the device in an illustration on page 171. Recoveries claimed for the washer include an efficiency per-

Dry blower or washer for concentrating gold gravels. As shown, it is being operated by W. H. Harris, the inventor

formance of practically 100 per cent in treating a placer gold ore averaging 30 colors to a cent; another recovery of 93 per cent on a gold ore that averaged 600 colors to a cent; 85 per cent on a flour gold ore of 1,000 colors to a cent; 95 per cent on a cinnabar ore; and 90 per cent on a tungsten ore.

The washer is made of ¼-in. 3-ply paneling, reinforced at the corners. Diameter of the drive wheel is 28 in., and that of the fan pulley is 2 in. The six-bladed fan has a diameter of 16 in.

Baffle plates under each riffle direct the air current onto the riffles. A trough in the bottom of the fan box catches the fine gold and discharges it into a pan attached to the side of the machine. The ore falls onto the riffles after passing through a 50-mesh screen, and any gold that may be blown up off them hits this screen and falls back again onto the riffles. In moving the device, a bicycle wheel is mounted on the upper end, and, with the adjustable legs serving as handles, it is then pushed in wheelbarrow fashion.

Mr. Harris states that this small hand-driven model will treat in a day as much gravel as one man can shovel onto the ¼-in. grizzly. A counterbalance and ball bearings reduce to a minimum the effort required in turning the drive wheel. The dust problem frequently encountered with similar machines is avoided by the use of a dust deflector.

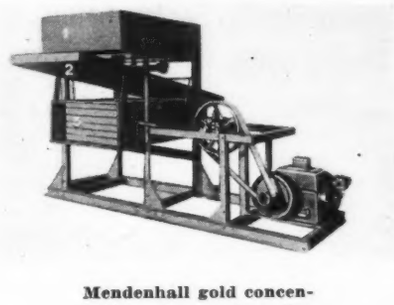
A larger dry washer having two separate fans and two sets of riffles and fitted with ball bearings so that it may be operated by hand or power has more recently been completed by Mr. Harris. The upper fan receives the gravel first. The upper riffle discharge is received on the second set of riffles. The unit weighs but 75 lb.

For Concentrating Gold Sands

Another recently developed machine for concentrating gold-bearing gravels and sands is the Mendenhall gold concentrator, made by the Mendenhall Gold Concentrator Company, 11 South La Salle St., Chicago, Ill. It is shown in a cut. The prime objective is to save the fine gold, which it does successfully, according to the company. The device consists of a water tank (1), a screen (2), and a series of superimposed trays (3), as shown in the illustration. By means of an oscillating action, the trays are caused to discharge off the highest points one into the other. Capacity is said to be 3 to 6 cu.yd. per hour. The machine weighs 1,100 lb., including a 1½-hp. engine, and occupies a 4x8½-ft. space. It is said to use but little water.

Gas Flasks of Light Metal

Flasks made of light metal are being used as containers for gas that is handled under high compression. They are made of alloys having a specific gravity about a third that of steel and which equals Siemens-Martin steel in tensile strength and elasticity. They are produced by Reinische Metallwaren u. Maschinenfabrik, Düsseldorf, Germany.



Mendenhall gold concentrator, for saving fine gold. (1) Water tank; (2) screen; (3) trays

Flasks of 0.8 to 5 liters' capacity and capable of standing 150 atmospheres' operating pressure (225 atmospheres' test pressure) are used for portable oxygen-breathing apparatus, and for holding carbon dioxide and compressed air in aviation work. Flasks with a capacity of 18 liters (largest diameter 152 mm.) are also made. The saving in weight is about 50 per cent.

Portable Air Compressor

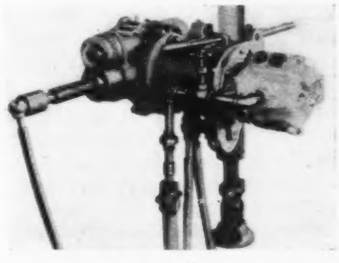
A new portable air compressor, developed by Ingersoll-Rand Company, New York, is of two-stage air-cooled design. By virtue of the improvements made, it will deliver 23 per cent more compressed air than previous models, size for size, according to the company. It has three cylinders, two of them low pressure, which are arranged in a V, and the third high pressure, in a vertical position between the two others. The superior efficiency of two-stage compression gives the unit an advantage at high altitudes and in hot climates. Adoption of air cooling, moreover, eliminates danger of freezing and keeps size and weight down. The partly compressed air from the low-pressure cylinders passes through an intercooler before going to the high-pressure cylinder. A fan draws air through the intercooler network. The air, thus cooled, enters the high-pressure cylinder at a temperature only a few degrees higher than the prevailing atmospheric temperature. The temperature of the air at the compressor outlet is 200 deg. cooler than that from the water-cooled single-stage machines. The unit is driven by a Waukesha, four-cylinder, heavy-duty gasoline engine. The new compressor is made in four sizes—namely, 125, 185, 250, and 370 cu.ft. per minute.

Magnesium Alloys Used in Core Drill

An extra light core drill for prospectors has been developed by E. J. Longyear Company, Minneapolis, Minn., with the aid of magnesium alloys and other features of construction. It is known as the Prospector and is shown in a cut. It is adapted for operation by either air, gas, or electric power and may easily be carried to remote locations. The drill head weighs 79 lb. Equipped with an air motor, the drill has a total weight of 177 lb. Despite its lightness, it is strong and efficient, the company claims. Its rated capacity is 250 ft. of Prospector ⅝-in. core or EX (⅞-in.) core.

Lightness has been secured by using magnesium alloys in nearly all stationary parts and by eliminating all metal not needed. The feed screw is of steel tubing, slightly smaller than the standard E screw, to save weight. No hoist is provided, for the same reason. For the depth of holes for which the drill is designed, the rods may be hoisted by hand, the company claims.

Owing to its lightness and convenient size, the air or electric model is said to be well adapted for use underground in limited working space. It will drill horizontal crosscut holes in a 5-ft. drift and vertical holes in any ordinary mine opening. One man can tear down, move 100 ft., and set up again in 15 to 30 min. Similarly, for surface work, the



The light weight of this diamond core drill has been obtained in part by using magnesium alloys in stationary parts

gas model, with its light tubular steel frame, is adapted for preliminary exploration work.

The air motor referred to is a standard, four-cylinder, reciprocating unit, which runs in oil in an aluminum housing. It drives the spindle through a special high-strength alloy pinion and gear. Air consumption at maximum speed is 70 cu.ft. per minute with 70- to 80-lb. air. The gasoline engine mentioned is a 4-cycle, 2-cylinder 5-hp. unit.

Structure Drilling on the Mesabi

One of the accompanying cuts show the old and the new bits used for structure drilling at the Canisteo pit at Coleraine, Minn., on the Mesabi Range. A description of structure drilling on the Mesabi was presented in *Engineering and Mining Journal* some months ago

Bits used in structure drilling on the Mesabi

by D. M. Gleason. The new bit is made by welding a regular 1¼-in. cross bit to a short piece of extra heavy 1¼-in. pipe and is an improvement over the old one, as it is easily made, drills as fast as the old bit, and passes the cuttings up through holes formed by the flutes of the cross bit. The new bit can be made in any mine shop and is easily tempered. The old bit required tempering equipment found only in larger shops.

Sealing Porous Castings

A new process for sealing porous or sweating castings has been developed by the Bakelite Corporation. It consists in forcing a special sealing liquid into the pores at an air pressure of 250 lb. per square inch until the liquid has completely penetrated the weak section of the casting. This is followed by baking for 1 to 2 hr. at 275 to 300 deg. F. The treatment may be applied to castings made from aluminum, nickel silver, brass, bronze, and similar materials. When properly treated the solution cannot be melted or dissolved; it is unaffected by cold water, by steam, and by temperatures up to 400 deg. F., and is resistant to most chemical reagents.

Remote-Control Speed Increased

Selection and operation of the control circuits of remote-control apparatus in less than one second's time is possible by a new supervisory control known as Polaricode, developed by Westinghouse. The exceptional speed is accomplished by a new principle known as "direct selection," which eliminates loss of time inherent in previous selective supervisory systems operated by either coded impulses or by synchronous selection. The new system is available for use on two- and four-line wires, the former controlling up to 25 remote apparatus units and the four-wire type up to 2,500 units, all at the same speed.

Variable Speed Reducer

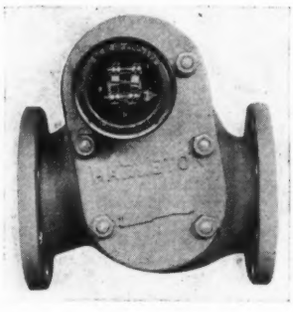
A new infinitely variable speed reducer, known as the Johnson, and ranging in capacity from 1 hp. up, has recently been developed by the Smith Power Transmission Company, Cleveland, Ohio. It is self-contained and runs in a bath of oil and will give infinite speed control from zero to any desired speed. The control is visible and quickly manipulated.

A Unique Motor

A motor into which is built fully 100 hp., despite its diameter of less than 12 in., has been introduced by The Louis Allis Company, Milwaukee, Wis. It is said to be ideal for direct-mounted grinding wheels, saws, cutter heads, and the like, where the diameter and peripheral of the cutting tool must be held to a minimum.

Controlling Centrifugal Pumps

A new piece of equipment for facilitating the automatic control of centrifugal pumps has just been introduced by Barrett, Haentjens & Company, Hazleton, Pa. It consists of a check valve (Type M) equipped with a Mercoid switch. One of these valves is shown in a cut. The purpose of the switch is to shut down the pump if it loses its water while running. The switch is connected into the low-voltage coil of the motor starter and is actuated by the valve flap. When the pump is delivering water, the flow through the valve raises the switch and closes the switch. When the pump loses its water, the flap falls, and the switch opens and stops the pump.



Check valve equipped with Mercoid switch for facilitating the automatic control of centrifugal pumps

BULLETINS

Compressors. Worthington Pump & Machinery Corporation, Harrison, N. J. Bulletins W-850 S1, S2, S3, S5, and S6, covering portable compressors of 120- to 360-cu.ft. capacity.

Rock Drills. Worthington Pump & Machinery Corporation, Harrison, N. J. Bulletin W-1201 S1B on horseshoe and sleeve-valve type rock hammers; also W-1202 S2A, on valveless rock hammers, screw-feed drifters, and sinkers; Bulletin W-1202 S1A, on horseshoe valve-type wet and dry screw-feed drifters; and Bulletin W-1204 S1A, on pneumatic-feed drifters for line drilling.

Motors. Westinghouse Electric & Manufacturing Co., East Pittsburgh, Pa. Bulletin on Gearmotors (electric motors having a built-in speed reducer), ½ to 75 hp. Pp. 8. Also a bulletin on multi-speed Gearmotors, ½ to 15 hp.

Motors. Falk Corporation, Milwaukee, Wis. Bulletin 261 on the Falk geared-head type Motoreducer (combination of motor and gear unit). Pp. 4.

Screens. Productive Equipment Corporation, Chicago. Bulletin 133. Heavy-duty, safety Jigger vibrating screen.

Gold Washing. Denver Equipment Co., Denver, Colo. Bulletin 3315. The Denver mechanical gold pan. Pp. 4.

Air Compressors. Sullivan Machinery Co., Chicago. Bulletin 88-G. Portable compressor, Wk-314, 505 cu.ft., V-type. Pp. 4. Also Bulletin 88-I, covering 8-in.-stroke angle compressors, direct-connected and belt-driven. Pp. 4.

Pumps. T. Shriver & Co., Harrison, N. J. Pamphlet. Shriver diaphragm pump. Pp. 6.

Corrosion Resistance. Stebbins Engineering & Manufacturing Co., Watertown, N. Y. Pamphlet. Linings for industrial equipment. Pp. 8.

Reducers. Allis-Chalmers Manufacturing Co., Milwaukee, Wis. Bulletin 1164. Motor reduction units. Pp. 4.

Flow Signal. New Jersey Meter Co., Philadelphia, Pa. Bulletin FS on "Flo Sig," an electric signal flow-indicating and controlling instrument. Pp. 4.

Belting. E. F. Houghton & Co., Philadelphia, Pa. Catalog. Vim Tred leather belting. Pp. 15.

Catalog-Directory. The Mining Department of the Los Angeles Chamber of Commerce has published a 200-page reference book entitled "Mining Directory-Catalog and Engineers' Handbook of Machinery, Equipment and Supplies for Mines, Mills, Smelters and Quarries." The first 83 pages are devoted to information, tabular and otherwise, that should be useful to anyone engaged in mining work. In particular, the text of the U. S. Mining Regulations, as it appears in Circular 430 of the U. S. Land Office, is included. Then follows a 73-page advertising section, succeeded by a 44-page classified index of products of Los Angeles County, California, for mining and quarrying, milling and smelting work. Price $2.

TRENDS AND DEVELOPMENTS IN THE INDUSTRY

NORTHWEST

Coeur d'Alenes Developments. Hecla Mining, having completed one major development, is starting another. The completed project is a raise that involved three years' work. The new work is an extension of the Hecla winze from the 2,800-ft. level to an additional 400-ft. depth, a new depth record in the Coeur d'Alenes. When Hecla in development work several years ago reached the 2,000-ft. level, a long crosscut was run, and at 2,500 ft. from the shaft a vertical winze was started near an important orebody. This by stages has been sunk 800 ft. When it reached the present level of 2,800 ft. from the surface workings, a crosscut was run back to connect with the main shaft, which was sunk the necessary 800 ft. to reach it. This winze is to go 400 ft. deeper. A crosscut from the new 3,200-ft. level, when reached, may be run back 2,500 ft. to the shaft. The other project, completed this week, is a raise from the 2,000-ft. level of the Hecla in the Star ground. This opens the Star by a raise of more than half a mile, reaching the old Moffitt tunnel level in the Star. It is far into the mountain from the Hecla shaft, and for that reason what is the 2,000-ft. level in the Hecla became the 4,000-ft. level in the Star. . . . Golden Center Mines has acquired three gold properties in the Granite district of Oregon—the Independence, Cougar, and Magnolia, all of which will be put in physical condition for operations immediately. R. B. McGinnis will be in charge of operations. Mr. McGinnis and Amor F. Keene report that the properties contain substantial reserves of proved and partly proved gold ore. Their estimate is that it will require about $155,000 additional money for active operations. This expenditure will include the cost of remodeling and enlarging the mill. . . . Liberty Gold Mining reports the purchase of property in the Swauk district at Liberty, 25 miles from Ellensburg, Wash. The group of four claims is adjoined on the west by the Jordin property, the scene of a gold strike early in the winter, and on the east by the Verdin mine, from which $46,000 was produced by its owner with an arrastre. Production continues from the Jordin. . . . Cutting of three veins in a tunnel driven 1,000 ft., near Murray, Idaho, is reported by Four Square Gold. In exploring the veins, the second, where crosscut, at 650 ft. from the portal, is 4 ft. wide and virtually all quartz, averaging $15 gold per ton. Associated with the quartz is much scheelite. The third vein, cut at the 850-ft. point, contains 8 in. ore, the greater part quartz that yielded $24 gold per ton and 3 ft. that averages $3.40 per ton, according to report.

Beryllium in Idaho. Latah County, in northern Idaho, may be an important commercial source of beryllium, according to the University of Idaho School of Mines. The mineral beryl has been found in considerable quantities. The ore occurs in mica pegmatite veins, associated with quartz, feldspar, and muscovite mica. Beryl has long been a byproduct in the mining of feldspar and mica, but with a demand for beryllium, the Idaho deposits might reverse the process, and produce beryl, with feldspar and mica as byproducts. Cost of extraction of beryllium from its ores is almost prohibitively high. Until this can be reduced, its commercial utilization will remain limited. Many laboratories are working on the problem, including the Idaho School of Mines.

The Latah County pegmatites in which beryl occurs were described by Dr. Alfred L. Anderson, of the Idaho School of Mines, in the February issue of the "Journal of Economic Geology." The beryl in these deposits forms crystals of pale greenish hue, up to 1 ft. long. No crystals of gem quality were found. The green gem beryl is the emerald, the most costly of all gems. Dr. Anderson's article treats of the origin of the minerals in the pegmatite veins, and explains why such minerals as beryl and commercial mica are so comparatively rare, although pegmatite bodies with quartz and feldspar are common. He also points out the practical applications of these findings in the prospecting for commercial pegmatite bodies and their exploitation. Valuable suggestions are included for prospectors searching for minerals containing such rare and valuable elements as lithium, columbium, tantalum, tungsten, and tin, found most generally in pegmatite veins similar to those in Latah County.

SOUTH DAKOTA

Construction Work in Progress. Homestake Mining, operating at Lead, has started work on the sinking of a 14x19-ft. new shaft, with two skip compartments, one cage compartment, and one compartment for accessories. A Nordberg-General Electric hoist will be used. The work of adding to No. 1 sand plant is proceeding satisfactorily. Foundations for the new 44x12-ft. leaching vats, of redwood, have been completed, and other equipment has been placed in position.

COPPER COUNTRY

Copper Mining Continuing. Calumet & Hecla and Copper Range, in the Michigan copper district, will continue to operate on the present basis, thereby providing as much relief as possible to employees. Calumet & Hecla operations are confined to upper levels in the conglomerate department, the richest ground. Shaft pillars and backs of old stopes are being mined. The newest workings, now idle and no longer kept unwatered, are below the 81st level haulageway and to the north of the Red Jacket shaft at depth. All amygdaloid shafts are closed. Calumet & Hecla has several years more of work before available copper ground in the upper conglomerate levels will be exhausted. Copper Range is working only its No. 4 Champion shaft. A drift also is being extended to the Globe property, in which an encouraging showing was disclosed by diamond drilling. Ground in No. 4 shaft is averaging well, and new openings are revealing ore of the same encouraging character. Most of the copper produced is being taken by the Hussey company, fabricators, of Pittsburgh, controlled by Copper Range.

War Department and the Copper Stocks. The copper surplus in this country, roughly estimated at one thousand million pounds, is holding back the domestic copper mining industry by keeping the price of the metal down to a figure well below actual cost of production. A large part of the surplus stocks, it is contended, was dumped into the country by foreign producers before the 4c. tax was imposed by the last Congress. Domestic production, kept at the lowest minimum in many years, is held to be well below that of consumption. Consumption, however, cannot make much inroad into the surplus for a long time to come. To save the industry, on which many thousands of American citizens depend for their livelihood, the suggestion has been made that the government buy the surplus stocks and turn them over to the War Department, as an investment. At 5c. per pound, this would require $50,000,000, which, it is contended, would be profitable, as copper, unlike cotton or other goods, is imperishable and the price ultimately will increase. The government has helped the railroads and the banks. The copper industry, also of vital importance to the country, is in equally dire straits. Government purchase of surplus copper also is urged as purely a preparedness measure. When the United States entered the World War, copper increased in price, so great was the need, and it finally was

stabilized at 26c. per pound. The mines were called upon to produce to the limit of capacity, which, incidentally, was the cause of much of their subsequent trouble, as the Armistice left them with large stocks of high-priced copper on hand, in the face of a declining market. The government and the industry would be in a favorable position, in the event of a national emergency, if surplus copper were taken over by the government now. To buy the metal at the present price would be economical. The sponsors of the plan claim that the copper industry would be stabilized thereby, for it then would receive a fair and reasonable price, in a market governed solely by the normal operation of the law of supply and demand. The mining companies would get cash enough to pay much of their indebtedness to the banks, thereby helping the financial institutions, and could re-establish themselves on a sound basis of operation. This, it is stated, would put labor back into the mines, mills, and smelters, bring relief to thousands of citizens, and aid in general recovery. . . . Copper producers are hopeful of favorable tariff action by the new Congress. The present 4c. duty on the metal is part of the internal revenue bill and expires by limitation June 30, 1934. It has had no appreciable effect, in so far as the prosperity of the copper industry and the communities dependent upon it is concerned, for the reason that copper has not been in demand. Actually, however, it has been effective in keeping out foreign copper, which, in itself, will hasten the recovery of the domestic industry when business improves. Owing to low cost of production of foreign copper, due to richness of ores, easy extraction, cheap labor, or to the recovery of copper as a byproduct with little or no expense chargeable to the metal, the domestic industry cannot survive without a tariff on copper importations.

ARIZONA

New Mill and Placer Operations. Akron Gold Mining expects to install a mill at its Jaeger mine, in the Comobabi district near Tucson. The mine was obtained in August, 1932, by purchase of options held by Gus Williams, of Globe, and E. R. Rice, of Miami. Williams and Rice had deepened the shaft on the property from 106 ft. to 140 ft., and in doing so had struck a ledge from which they shipped several carloads of gold ore running about $50 per ton. Since taking the mine over, the Akron company has deepened the shaft to 230 ft., and has done 350 ft. of drifting on the 150- and 180-ft. levels. A car of ore is shipped to the smelter about every ten days. The shaft makes 7,000 gal. water daily, which is available for milling purposes. Herbert H. Huntsberry is in charge of operations. . . . A syndicate, composed chiefly of Phoenix men and headed by R. E. Grace, of that city, has taken a bond and lease on the Bunker Hill gold and silver property at Tombstone. The property was operated practically continuously from 1915 to 1931 and was a Phelps Dodge branch from 1925 until disposed of to the syndicate. It was idle during 1932. A stock company is to be formed to take over the holdings of the syndicate, development work is to be started, and several leases will be issued. . . . The placer property known as the Cherry Minnehaha, near Waggoner, in Yavapai County, is being sampled by Operators & Developers, of Prescott, for North American Mines Exploration. Test pits are being sunk to bedrock at the corners of 200-ft. squares. The samples from the pits are carried by burros to Minnehaha Creek, where the gold is removed in sluice boxes. The black sands, after the free gold has been removed, assay about $1 per ton. Sampling is in charge of Clyde Mathews for Operators & Developers, and is being checked by Carl Barth for the North American company. . . . A mill for gold-ore treatment is being erected at the Morgan-Knight property, 50 miles east of Ajo, on the Tucson highway. The principal working on the property is a 300-ft. inclined shaft. Six carloads of ore have been shipped to El Paso. Keith Knight is manager. The mill is being installed by Miles M. Carpenter, of Tucson.

NEVADA

Activity in Rosebud and Scossa Districts. Frederick S. Parkhurst, Jr., has taken over the Surprise claim, at Rosebud, in Pershing County, on a working agreement with the owner, Mrs. Cora Willig Bishop, of Sulphur. The property adjoins the Ragged Six lease, on the Brown Palace mine, which recently shipped a 50-ton car of ore to a Utah smelter, estimated at $40 per ton. . . . At Scossa, 6 miles from Rosebud, W. P. Frick and W. F. Gray have purchased an interest in the Scossa Red Top and have pledged funds for equipment and development of the property. . . . The flotation mill of Recovery Milling at Silver City, on the Comstock lode, is making a test run from ore at the Overman shaft of Consolidated Chollar, Gould & Savage. This is the first custom mill on the Comstock to adopt flotation. J. L. Dobbins, of San Francisco, general manager, bought the McTigue amalgamating mill, discarded the plates and vanners, and installed a ball mill and six Kraut flotation machines, five roughers, and one cleaner. The ore, crushed in three five-stamp batteries, passes to a classifier in closed circuit with the ball mill. The jaw crusher and stamps are operated by water power; the ball mill, classifier, and other machinery, by electric power purchased from Sierra Pacific Power. Three other custom mills at Silver City —the Trimble, Donovan, and Overland —employ amalgamation, in batteries and on plates. . . . Hydraulic mining commenced March 13 for the season at the Round Mountain placer mine of Nevada Porphyry Gold, with one giant operating eight hours. Cold nights have checked the water run-off. Continuous operations will be conducted when a full head of water is available. Recovery averages $1 per cubic yard. . . . Power jigs have been installed at the Guild-Platt placer mine in the Singatse range near Yerington, Lyon County, to recover gold, which escapes the riffles in the sluice boxes, from black sands. . . . The Nevada legislature has enacted a law authorizing county commissioners to issue prospecting permits on patented mining claims acquired for delinquent taxes. Only prospectors with one year's actual work in Nevada are eligible, and permits are limited to two claims. Any time within six months after receiving a permit, the prospector may purchase the claim or claims from the county for the amount of taxes and accrued costs. . . . The mine, mill, and cyanide plant of Gold Hill Development, 5 miles north of Round Mountain, have been closed down. The workings struck a barren zone at 500 ft., and a winze was sunk to the 600. A crosscut on that level cut the vein but failed to strike an oreshoot. A heavy flow of water was encountered on the 600 level, which would necessitate installation of a large pump. Gold Hill was financed and operated by the Tonopah Mining and Tonopah Belmont, and stockholders of those companies. The mill has been in continuous operation since 1930.

Daisy Group Is Optioned. Good Hope Mining, operating on the Mother Lode of California and in Arizona, has optioned to M. J. Kearns the Daisy group of eight mining claims at Humboldt. Mr. Kearns plans to equip the property with mining plant and to erect a mill. The gold ore, in a dike 1,500 ft. long, has already been exposed by open cuts and 50-ft. shafts.

MONTANA

Gold Monopolizes Attention. Storey estate interests, of Bozeman, have installed a new dredge on placer holdings in Norwegian Gulch, near Norris, Madison County, and the clean-up is running about $2,800 per week. . . . G. Box, Charles Morris, and associates of Pony, Madison County, are operating the Strawberry mine, close to the Boss Tweed & Clipper group, near Pony. Commercial ore has been opened, the main values being in gold. The mill on the property was last run by Montana Metals, of which Ben H. Jackson was the head. . . . At the annual meeting of stockholders of Consolidated Mining & Milling, the financing contract recently made with California capitalists was approved. The contract provides enough money to put production at the Ida mine, near Boulder, on a paying basis. The new shaft is down 70 ft., and will be sunk an additional 130 ft. to tap gold veins that show milling ores on the surface. . . . J. W. Marchbank and Joseph Ferris, of San Francisco, expect to start operations soon on the Wilson Bar placer, in Nevada Creek, near Helmville, Powell County. According to engineers' reports, the ground contains about 16,000,000 yd. gravel, about 25 ft. deep. Several test holes have been dug, showing values of 50c. per yard. Mr. Ferris, who is credited with installing the first gold dredge in Venezuela, expects to install a floating dredge on Wilson Bar to handle 5,000 cu.yd. gravel per day. . . . I. B. Mining, operating at Bannack, Beaverhead County, is now running its mill and treating an average of about 120 tons gold-silver ore per day. Charles Hancock is manager and William Franke mill superintendent.

Cyanide Leaching Plant Planned. Little Ben Mining, operating the August mine, near Landusky, Phillips County, is contemplating the erection of a coarse-crushing leaching plant. Some good bodies of ore have been opened during the winter, with assays from $6 to $3,300 per ton, mostly in gold. . . . Butte Highlands Mining, operating under supervision of E. J. Scott, in the Highland Mountain, south of Butte, during the past six months has shipped 600 tons of ore, which netted about $10,000, after payment of all freight, treatment, and a 15 per cent royalty charge. Trucks are now removing snowdrifts, after which an accumulation of ore at the mine will be hauled to Rocker and thence to the Washoe smelter of Anaconda Copper. The southeast drift is in 150 ft. of gold ore. . . . Interstate Mining, incorporated in Washington, is taking over the Sleman Placer ground, on Little Pipestone Creek, Jefferson County, south of the main highway between Butte and Whitehall, 1¼ miles long and about one-half mile wide. Test holes show assays up to $1 per yard. As soon as weather conditions warrant, drag lines will be installed and operations started. . . . A group of Montana mining men have organized the Montana Silver Association, to further the cause of the white metal. Charles R. Brazier, of Helena, heads the organization.

CALIFORNIA

Placer and Vein Mining Activity. The Hope mine, near Sonora, once a noted producer, is being reopened by F. A. Reall. . . . Paramount Mining, operating the Centennial and Conrad mines, in the Ophir district, reports a second strike of high-grade ore. Considerable development work is being carried on from the 300-ft. inclined shaft. . . . Preliminary work, including the constructon of camp buildings and erection of machinery, has been started on the Punch Bowl placers, in the vicinity of Jamestown, by Los Angeles interests. . . . Purchase of the New York mine, in Plumas County, has been announced by Yellow Jacket Consolidated, of Los Angeles. The property is equipped with underground and surface mining machinery, including a mill, and a considerable tonnage of $7 ore is said to be available for milling. . . . The famous Roxbury placers, in the Yreka district, are being rehabilitated, and machinery will be installed for early production, by the Day Syndicate, of Wallace, Idaho. More than twenty men are employed at the King Solomon mine, near Cecilville, where a new mill and flotation plant have been placed in operation recently. . . . Rehabilitation work has been started at the old Sawyer mine, in Kernville, by W. A. James and the Matlock brothers.

Beebe Operations Resumed. Atolia Mining, operating the Beebe and Eureka-Woodside mines, at Georgetown, has resumed milling operations after a brief shutdown, during which changes were made at the mill to increase capacity. . . . A diesel power plant and additional pumping equipment are in course of erection at the property of Golden Center Mining, operating in Grass Valley. Idaho-Maryland Consolidated reports the completion of the new compressor building, and the erection of a new compressor and several auxiliary units. . . . The Gladstone mine, at French Gulch, credited with a production of $4,000,000 in sixteen years, is being reopened with G. T. Salsbury in charge of operations. A complete plant of modern equipment is to be installed. . . . Death Valley Gold Mining will build a 70-ton flotation mill at its property in the Yucca Grove district, to handle ores now being stockpiled. . . . Reconditioning of the mill at Carson Hill mines is rapidly nearing completion, and large-scale production is scheduled to start soon. Extensive deposits of low-grade ore are available for milling. . . . Empire-Star Mines has purchased the Sultana mining properties, which adjoin the company holdings on the southeast.

Revival of work at the old camp of Adin, Modoc County, is reported. C. E. Whiteman has taken a bond on the Juniper and is sinking on a three-foot vein. He has installed an Ellis ball mill, amalgamation plates, and a rocker to treat the old dumps and ore from current work. The gold is fine and the recovery good. Stratton and Stratton, of Wallace, Idaho, have leased the Blue Bell and are retimbering the 200-ft. shaft. The Golden Eagle has been bonded to William Nurral, who, it is rumored, will install cyanidation equipment to treat the dumps.

MEXICO

Mineral Output Shows Decline. Reduction in Mexico's metal and mineral production is indicated by Ministry of National Economy reports, which show a marked drop during the 1932 January-September period, compared with the first nine months of 1931. The figures for the two periods, 1932-1931, respectively, in kilograms, are as follows: Gold: 13,762, 15,114; silver: 1,552,008, 2,079,869; copper: 27,264,960, 43,659,426; lead: 107,721,269, 185,104,997; zinc: 40,485,603, 103,066,362; antimony: 1,035,637, 4,193,841; amorphous graphite: 1,514,435, 2,692,789; arsenic: 3,172,148, 5,639,465; cadmium: none, 31,831; tin: 591,344, 545,266; coal: 508,175,438, 616,048,306. The department also reports an output of 5,064 kilos molybdenum; 360 kilos selenium; and 2,573 kilos bismuth during the first nine months of 1932. . . . Difficulties the Cia. Minera y Explotadora de Amparo, operating in Guanajuato state, has had with its employees have been settled to the satisfaction of both parties as the result of a conference in Guanajuato City of company and workers representatives, at which an agreement was made that the enterprise will pay indemnities to workers for dismissals and injuries from accidents. . . . Mexican mining interests operating near the border are feeling the pinch of the money situation in the United States. The first of these enterprises to be affected, dependent on United States funds to carry on the work, is Victoria Mining, operating near Nogales, Sonora state, which has been authorized by the state labor board to suspend operations temporarily. . . . Activities have been resumed by Naica Mining on its property near Naica, Coahuila state. Several carloads of ore weekly are being shipped to the refinery in Monterrey, Nuevo Leon state, operated by Peñoles Mining, according to the Ministry of National Economy. . . . More than 500 prospectors and unemployed miners are obtaining a good living by working a gold field that has been discovered in the Cerro de la Concha region, near Talamantes, Valle de Allende, Chihuahua state, according to members of the federal chamber of deputies from that commonwealth.

CANADA

Economy from Good Ventilation. International Nickel's annual report for 1932 records a steady increase in the number of uses for nickel, especially in the iron and steel industries. Although net earnings for 1932 were less than half those of 1931, the surplus account has been kept up to the substantial amount of $14,688,559 by omitting dividends on common stock. The company's Frood mine is equipped with a complete ventilation system, the economy of which is indicated by the following passage from the report: "No. 1 ventilating shaft was completed to the 3,100-ft. level and has made pronounced improvement in underground air conditions. The average temperature is now 67 deg. Fah., compared with 78 deg. prior to the installation of the new system. Relative humidity has been decreased by 10 per cent. The fact that the amount of ore per man mined in stopes has increased by about 36 per cent, and that there has been a decrease of about 60 per cent in lost-time accidents, is largely due to improved ventilation." The Frood is the pioneer among Canadian metal mines in proper ventilation. The system was installed primarily to prevent fires, but it would seem to be justified by the increase in labor efficiency alone.

Noranda's Position Favorable. Noranda is nominally a copper mine, but over two-thirds of the value of metals recovered in 1932 ($11,752,628) were in gold, preponderance of which is due partly because of copper production restriction, and in part to the increasing use of concentrating ore containing proportionately little copper and high gold. The statement of ore reserves above the 2,475-ft. level, however, indicates the swing throughout the mine as a whole from copper to gold. Development of the known orebodies above the 1,975-ft. level yielded more ore than was smelted, and the reserves have thus been increased by the amount developed between the 1,975 and 2,475 levels. . . . The Larder Lake field, 25 miles east of Kirkland Lake, was the scene of the first gold rush in that part of Ontario, 25 years ago. A recent discovery in Hearst Township in the heart of this area is a shear zone in porphyry with good ore distributed across economic mining widths. Substantial exploration companies hold the ground, and rapid development is assured. . . . Howey Gold Mines made an operating profit of $470,412 during 1932. The annual re-

port states that mill capacity per day will be increased this year to 1,300 tons, effecting an estimated reduction of operating cost to $1.85 a ton. Reserves of ore above the 1,000-ft. level will provide eighteen months' supply for the enlarged mill, in addition to which a large section, not yet sufficiently developed, must be included in the reserve. Partial development between 1,000 and 1,350 ft. has indicated 500,000 tons averaging $2.45.

New Mine and Mill Developments. Halcrow Swayze Mines is installing a mining plant to develop its vein system underground. Fairly good gold ore in substantial volume has been indicated by surface work. As in the case of the Kenty property in the same area, the management hopes to prove rapidly a large tonnage of payable ore by this means, the distribution of gold being considered too erratic to give reliable results by diamond drilling. . . . The Dorr Company in Toronto has contracted to install the 50-ton gold mill of Central Patricia Mines. The mill is expected to be in operation by June. Part of the supplies and equipment have been delivered by airplane, part by a 120-mile winter road. . . . Siscoe Gold Mines in 1932 milled 63,998 tons, yielding $15.72 a ton. Mill capacity is being increased to 300 tons per day. . . . The Little Long Lac gold property, 80 miles north of the Lake Superior shore and close to the Canadian National Railways east of Lake Nipigon, is being prepared for rapid underground development. Surface showings and diamond drilling have shown a vein about 1,000 ft. long, in which is indicated a shoot of ore 400 ft. long, of stoping width and of high grade. Four hundred tons of machinery and supplies has been delivered at the property, and a shaft has been started with an objective of 500 ft.

BRITISH COLUMBIA

Mining Law Revision. Changes of an important character in the mining laws of the Province, although in no way affecting the underlying principles, will be submitted to the Provincial Legislature during the present session. The amendments, affecting the Mineral Act and the Placer Mining Act, are in the main designed to bring those statutes more in line with present conditions. In the past the provision imposing the severe penalty of forfeiture of title for failure to renew the Free Miner's Certificate on the due date has in numerous instances caused considerable hardship where certificates have lapsed on account of illness or other unavoidable causes, or even in consequence of the deliberate dishonesty of agents or partners. To afford relief in this connection both acts are being amended to provide for a period of six months' grace after the lapse of the certificate for its renewal, during which period title will be protected. Other important amendments to the Placer Mining Act aim to clarify and simplify the conditions under which leases can be acquired and will greatly facilitate location and surveying. Hereafter there will be only one class of lease with a maximum area of 80 acres, which will be located in a similar manner to mineral claims. In the Mineral Act, provision is being made requiring that metal identification tags be attached to location posts; and, in future, mounds or cairns of stone may be legally used instead of wooden posts, to facilitate location of mineral claims above timber line.

Placer-Mine Developments. Recent rich discoveries on Otter Creek, in the Atlin district, by the Cie. Française des Mines d'Or, may have far-reaching consequences as affecting the future of placer mining in this district. The company, which for the past three or four years has been engaged in developing its leases on this creek for hydraulic operation, has only recently attempted to reach bedrock by sinking. The difficulties of sinking, owing to the character of the ground and the inflow of water, were considerable, but were surmounted. Bedrock being reached, extraordinarily rich gravel, yielding from $15 to as high as $45 to the yard, was encountered. The important point is that throughout the district only surface gravels that have yielded but modest profits have been mined heretofore, no successful attempt having been made until now to gain bedrock. The results on Otter Creek therefore are indicative of new possibilities for successful exploration in this connection and will have a stimulative effect on mining in the Atlin district. . . . The N. A. Timmins Corporation, which has been developing a small gold property, the Surf Point, on Porcher Island, is now erecting a flotation plant. The ore, an auriferous pyrite, is of high milling grade and occurs in narrow veins in the granite. Premier Gold Mining has acquired the Wonder group of claims, in the Cedar Creek area of the Cariboo district. In addition to the operations in this district of Cariboo Gold Quartz, whose pilot mill is now running steadily, development work by Cariboo Consolidated Mines is proceeding on Island and Proserpine Mountains. On the first-mentioned group, a 2,000-ft. tunnel is being driven; and on Proserpine Mountain an old shaft and workings have been unwatered and a new crosscut tunnel to intercept the orebodies at a somewhat lower horizon has been commenced. The Britannia company is also driving a tunnel on its property at Barkerville.

Milling Developments. Recent developments on the levels below the main adit on the Bralorne mine, in Bridge River district, afford satisfactory evidence of the continuation of the orebodies at depth, the extension of the King vein having been opened up beyond a fault. The capacity of the mill has been increased to 125 tons a day. . . . Erection of a 100-ton mill on the property of the Big Slide Mining Company on Pavilion Mountain, in the Lillooet district, is contemplated. . . . Nicola Mines & Metals has acquired the Planet mine, in the Nicola district, from the company that formerly operated it and which is now in liquidation, together with the adjacent Donaghue property. Development operations will start soon. Investigation of the commercial possibilities of the Fairview district, one of the oldest camps in the Province, but unproductive for the last 30 years, is now proceeding and has included the unwatering of the old Morning Star shaft, a property typical of the large occurrences of low-grade quartz characterizing the camp. The possibilities here depend upon the delimitation of available reserves for the maintenance of large-scale operations, the orebodies being admittedly low grade.

AFRICA

Randfontein Plans Vast Expenditures. Randfontein Estates is to spend £700,000 for additions, to bring capacity to 10,000 tons per day and to improve mill extraction. An increase in capacity of 500,000 to 600,000 tons per annum is planned; £210,000 is to be used for enlargement and modernization of the reduction plant, and £500,000 on "other equipment and work incidental to enlarged scale of operations." In one recent month the company's profit was increased by £120,000 as a result of the higher price obtained for gold. The directors were thus enabled to expedite the expansion program to an extent that will insure the increased capacity being brought into effect within a year. The cost is to be spread over two years, so that shareholders need not be apprehensive as to the effect on dividends. At the annual meeting of the company the chairman said, "The mine has a very long life provided the grade of ore developed is maintained, and in this respect it is significant that values have improved in the lower levels."

Progress at Daggafontein. Among Far East Rand properties engaging attention are Daggafontein Mines and East Daggafontein. The latter company, on account of the gold premium, earned a profit of £47,251 in January, compared with £21,110 in December. At the No. 2 shaft of this mine, station cutting and development are proceeding, and the main pumping plant is complete. Almost all work is in waste rock, but in the vicinity of the station the reef is being exposed, and assays have confirmed former development results. . . . At East Daggafontein, floated a few months ago, exploratory work from the No. 1 shaft is commencing, and the north haulage from the shaft will enter the East Daggafontein property. The east and south haulages from the same shaft will be pushed ahead in accordance with the comprehensive development program.

Pegging Activity on Rand Extensions. The gold premium has stimulated activity and optimism in the Heidelberg and Potchefstroom-Klerksdorp districts. Continuation of the auriferous conglomerates of the Witwatersrand beds into these areas was established many years ago, and several companies and syndicates were formed. For the past twenty years the areas have lain fallow, but extensive pegging has recently taken place, especially in the southwestern extensions of the Main Reef. Consolidated Goldfields has acquired properties formerly belonging to the Western Rand Estate, to the southwest of Randfontein, and boring is now in progress. Activities in connection with prospecting rights and mineral options in the Potchefstroom

area are proceeding. Almost the whole country west of Welverdiend through Reismerbult on to the Buckingham, and then along the farms south from Dovesdale on the Bulskop, in an almost direct line with Eleazer, to Buffelsfontein, on the Vaal River, is under option to a Rand mining and financial group. Prices of land under option are being increased up to £15 per morgen (2.1 acres), and an independent company has been formed to develop Katdoornbosch and one or two adjoining farms.

Transvaal Gold Output. With a yield of 11,553,564 oz. gold during 1932, the Transvaal made another record. Since 1924, when the return was 9,574,918 oz., the output has steadily advanced. Total recorded production from the Transvaal since the beginning of 1884, and up to the end of 1932, is 272,591,713 oz.

Copper Production and Costs. Union Minière announces that, beginning with April, copper production will be increased to 56,000 metric tons per annum. For a year past it has been producing 40,000 metric tons annually. The depressed state of business in the Belgian Congo, mostly dependent on the activities in the copper center, made this expansion desirable. Union Minière enjoys preferential tariff treatment on the copper it sells in France; it is reported to have disposed of a considerable tonnage to the Japanese Government, and also is believed to be able to realize a small profit on its output even at the prevailing low price. The increase will enable the company to reduce production costs. . . . The Hon. R. M. Preston, a director of Rio Tinto, says in an article in *The Financial News* that "The Northern Rhodesia mines, favored by the divorce of sterling from gold, are producing copper at a cost which has never hitherto been approached. The larger of the two mines at present working (Rhokana) is reported to be now delivering blister copper in European ports at a cost of under £20 a long ton, equivalent, at current exchange, to 3c. per pound." The cost of producing copper in Northern Rhodesia was lower in January and February than in any previous period. Though the costs of these two months cannot be depended on to establish a general average, the ability to reduce them after they were down to the lowest on record, for copper not associated with precious metals or other valuable byproducts, is impressive. . . . Reports affirm that quantities of "fire-refined copper" were being offered in England and on the Continent. Strictly speaking, the only fire-refined copper in England is being produced by British Copper Refiners in its new plant at Prescot, Lancashire. That plant has been operating continuously and satisfactorily for three or four months, and is outputting about 2,000 tons monthly. The greater portion of its product is going to British Insulated Cables, to be drawn into high-conductivity wire. Roan Antelope and Braden are producing highly pure blister copper, which can be readily fire-refined, and a part of the product of Rhokana is almost equally pure. The report probably refers to "copper fit for fire refining," perhaps from the Braden mine, in Chile.

AUSTRALASIA

Commonwealth Flotations. New gold mining companies formed recently include: Burnside Gold Prospecting Syndicate, to prospect a property in the Northern Territory; New Year Gold, to acquire an option over a gold mine at Walhalla (Vic.); Wilson's Hill Gold, to operate at Marong, near Bendigo; Strachan Argyle Mines, to operate at Dark River, Vic.; Frank's Reward, to develop a find at Wannaway, W. A.

Victoria. At the Hercules mine, Bendigo, rich returns continue to be obtained from the new saddle reef below the 1,620-ft. level, which has yielded 8,841 oz. from 4,290 tons, and has made possible large dividends. Lower-grade ore is now being mined. Great activity is reported from the Bendigo district, and many mines are being unwatered. The State Government is erecting another 10-head battery to facilitate prospecting; 1,220 prospectors are at work. . . . Boring operations on the deep leads at Allendale have been completed. Three leads were discovered at depths of 120, 130, and 160 ft. Leases totaling 1,000 acres have been pegged, and the site for a main shaft has been selected. . . . While developing the Yankee mine, at Trentham, Gold Mines Development crushed 1,147 tons for 501 oz., which at the present price of gold is highly payable. The main shaft will be sunk further, and another level opened. . . . Sir Herbert Gepp, Dr. Frank Stilwell, and W. Baragwanath report to the Federal Government on the Maryborough gold field, Victoria, that the belts of gold-bearing slates from Talbot to Dunolly fully justify further investigation. Gold content is 4 dwt. per ton. . . . An electrical prospecting survey was recently carried out on the Cassilis gold field, where small rich and complex oreshoots occur along four parallel lodes.

Western Australia. A promising gold field has been found at Wannaway, 70 miles south of Kalgoorlie, over a strike length of 8 miles. Frank's Reward has proved a highly payable lode by shafts over a length of 280 ft.; and from a near-by lease 17 tons was crushed for 245 oz. The main shaft at Frank's Reward is at a depth of 50 ft., the lode being 3 ft. wide. T. Blatchford, Government Geologist, reporting on the field, stated that it had considerable promise, as extensive shearing had been disclosed and rich oreshoots may be found along the shear zones. Boulder Perseverance is meeting rich ore in a winze below the 1,100-ft. level of the Perseverance lode. At 63 ft. north from the bottom of the winze the ore is 13 ft. wide, assaying 36 dwt. . . . Western Gold Mines has been formed to acquire an option of purchase over the Mararoa gold mine, at Reidy's. Colin Fraser, chairman of Broken Hill South, is chairman of the new company, and Richard Hamilton, formerly general manager of Great Boulder, Kalgoorlie, and H. E. Vail, consulting engineer to Lake View & Star, are also directors. . . . Broken Hill Pty. is negotiating for the purchase of North Kalgurli, which during 1932 carried on an extensive program of underground development. . . . Great Boulder Pty. has met with several rich seams in a diamond-drill hole east from the 230 level at Robertson's shaft. . . . Under a working option, Broken Hill Pty. has been carrying out vigorous development on the Enterprise mine, at Kalgoorlie. Crushings made at the Boulder Perseverance mill have totaled 2,116 tons, yielding 1,478 oz. The Enterprise lease adjoins on the north the Great Boulder mine. No decision has been made regarding the option. . . . Lake View & Star has carried out 200 ft. of driving on the new lode recently found in a crosscut 300 ft. from the 1,400-ft. level drive of the Lake View lease. The lode is in virgin country, and the ore proved has been highly payable. The company has also commenced driving on the 1,200-ft. level. . . . Wiluna is gradually extending its milling plant, but some months will elapse before the full capacity of 40,000 tons monthly can be milled. During December the working profit was £32,095, and £20,173 was spent in construction. The second additional diesel engine has been placed in commission, and the power supply is now more than sufficient for full mill requirements. Rapid progress is being made in mill extensions and in the erection of a small plant to re-treat accumulated cyanide residue. The assay office was destroyed by fire recently and a new building is being erected. The company has repaid £75,000 of the £300,000 of 6 per cent seven-year notes guaranteed by the Commonwealth and State Governments.

Queensland. Mount Coolon, Queensland, which commenced operations recently, has cleaned up 3,150 oz. of fine gold from 4,833 tons for 27 days' milling. The main shaft has been deepened, and a new level is being opened. Three winzes are being sunk from the 300-ft. level. The company has sold for forward delivery about half of its output of gold for the six months ended June 30, 1933, at £6.1.9 sterling per ounce. . . . Mount Morgan concentrate is being shipped to Port Kembla, N.S.W., for treatment. By the end of March 1,800 tons weekly will be handled, and this will be further increased to 3,000 tons by the end of April. The mine appears to have a profitable future. Ore is won cheaply from a large open cut, and the company has available 664,270 tons assaying 6.5 dwt. gold and 0.9 per cent copper, and 8,000,000 tons assaying 4.4 dwt. gold and 1.72 per cent copper.

Tasmania. The Tasmanian Government is considering a proposal to reclaim by drainage 2,000 acres of osmiridium bearing swamp on the banks of the Adams River, Adamsfield. Where tested, this wash proved to be rich, and reclamation work on it would keep 200 men employed for several years. At present, 150 men are prospecting for osmiridium at Adamsfield, and with osmiridium at £8 per ounce they are barely making a living. The drainage scheme would involve making a water race almost a mile long to the Adams River falls.

New Guinea. The first clean-up from New Guinea Goldfields' new mill gave 1,157 oz. of gold and 449 oz. silver. No details of tonnage treated were made available. The mill is now operating with greatly improved recovery, and an annual profit of £200,000 from mining operations is anticipated.

Daily and Average Monthly Prices of Metals

March, 1933

United States Market

Mar.	Electrolytic Copper, Domestic Refinery	Electrolytic Copper, Export Refinery	Straits Tin, New York	Lead, New York	Lead, St. Louis	Zinc, St. Louis
1	4.775	4.675	23.500	3.000	2.875	2.650
2	4.775	4.625	23.600	3.000	2.875	2.675@2.700
3	4.775	4.625	23.800	3.000	2.875	2.675@2.700
4	4.775	—Bank Holiday—		3.000	2.875	2.675@2.700
6	4.775	—Bank Holiday—		3.000	2.875	2.700
7	4.775	—Bank Holiday—		3.000	2.875	2.700@2.750
8	4.775	—Bank Holiday—		3.250	3.125	2.825
9	5.275	—Bank Holiday—		3.250	3.125	3.000
10	5.275	—Bank Holiday—		3.250@3.350	3.125@3.225	3.050@3.100
11	5.525	—Bank Holiday—		3.350	3.225	3.100
13	5.525	5.000	24.125	3.350	3.225	3.175
14	5.525	4.975	24.250	3.350	3.225	3.175@3.200
15	5.525	4.925	24.350	3.350	3.225	3.200
16	5.275@5.525	4.850	24.500	3.350	3.225	3.150@3.200
17	5.275	4.850	24.450	3.350	3.225	3.150@3.200
18	5.275	4.825	24.450	3.350	3.225	3.150
20	5.025	4.775	24.050	3.350	3.225	3.150
21	5.025	4.750	23.950	3.250	3.125	3.150
22	5.025	4.750	23.950	3.100	2.975	3.150
23	4.775	4.750	24.200	3.000	2.875	3.100
24	4.775	4.800	24.500	3.000	2.875	3.100
25	4.775	4.775	24.500	3.000	2.875	3.050
27	4.775	4.775	24.500	3.000	2.875	3.000
28	4.775	4.750	24.250	3.000	2.875	2.950@3.000
29	4.775	4.700	24.450	3.000	2.875	2.950
30	4.775	4.700	24.550	3.000	2.875	2.900@2.950
31	4.775	4.700	24.500	3.000	2.875	2.900
Av. for month	5.011	4.779	24.221	3.146	3.021	2.987
Averages for Week						
1	4.775	4.663	23.308	3.000	2.875	2.640
8	4.775	.....	.....	3.042	2.917	2.719
15	5.442	.....	.....	3.325	3.200	3.123
22	5.171	4.800	24.225	3.292	3.167	3.158
29	4.775	4.758	24.400	3.000	2.875	3.029
Calendar Week Averages						
4	4.775	4.635	23.460	3.000	2.875	2.669
11	5.067	.....	.....	3.192	3.067	2.904
18	5.421	4.904	24.354	3.350	3.225	3.177
25	4.900	4.767	24.192	3.117	2.992	3.117

Silver, Gold, and Sterling Exchange New York and London

Mar.	Sterling Exchange, "Checks"	Sterling Exchange, "90-Day Demand"	Silver, New York	Silver, London	Gold, London
1	3.410000	3.440000	26.5000	17.0625	121s11½d
2	3.430000	3.460000	26.6250	17.0000	121s 7½d
3	3.450000	3.490000	27.2500	17.0625	120s 8d
4	—Bank Holiday—			17.1250	.....
6	—Bank Holiday—		29.7500	17.6875	119s 7d
7	—Bank Holiday—		29.5000	17.8750	119s 7d
8	—Bank Holiday—		29.3750	18.0000	118s 11d
9	—Bank Holiday—		30.0000	18.4375	119s 7½d
10	—Bank Holiday—		29.7500	18.1875	119s 2½d
11	—Bank Holiday—		29.2500	18.2500	.....
13	3.400000	3.400000	27.7500	17.7500	123s 2 d
14	3.445000	3.485000	27.5000	17.6875	123s 3 d
15	3.452500	3.490000	27.5000	17.5625	123s 3 d
16	3.462500	3.502500	28.2500	17.8750	123s ½d
17	3.462500	3.492500	28.0000	17.6250	123s 3 d
18	3.458750	3.488750	27.5000	17.3750	.....
20	3.438750	3.468750	27.3750	17.2500	123s 2 d
21	3.427500	3.447500	27.3750	17.4375	123s 4½d
22	3.426250	3.451250	27.2500	17.4375	123s 9 d
23	3.420000	3.440000	27.8750	17.6875	123s 6½d
24	3.431250	3.455000	27.3750	17.4375	123s 6 d
25	3.425000	3.445000	27.3750	17.5000	.....
27	3.415000	3.440000	27.2500	17.4375	123s10½d
28	3.412500	3.435000	27.3750	17.5000	123s 9½d
29	3.412500	3.432500	27.6250	17.6875	123s11½d
30	3.427500	3.450000	27.5000	17.5625	123s 5½d
31	3.420000	3.440000	27.2500	17.3750	123s 4 d
Av. for month	3.43138	.....	27.928	17.588	.....
Averages for Week					
1	3.41229	.....	26.646	.....	.....
8	.....	.....	28.500	.....	.....
15	.....	.....	28.625	.....	.....
22	3.44604	.....	27.625	.....	.....
29	3.41938	.....	27.479	.....	.....

Calendar week averages: New York Silver, Mar. 4th, 26.675 Mar. 11th, 29.604; Mar. 18th, 27.750; Mar. 25th, 27.438.

London Market

Mar.	Copper, Standard, Spot	Copper, Standard, 3 Mo.	Copper, Electrolytic (Bid)	Tin, Spot	Tin, 3 Mo.	Lead, Spot	Lead, 3 Mo.	Zinc, Spot	Zinc, 3 Mo.
1.....	28.0000	28.2500	31.7500	148.7500	149.0000	10.4375	10.7500	14.2500	14.5000
2.....	27.8125	28.1250	31.7500	147.7500	148.3750	10.4375	10.8125	14.1250	14.3125
3.....	27.4375	27.6875	31.5000	146.8750	147.3750	10.3750	10.6875	14.1250	14.3125
6.....	27.1250	27.4375	30.7500	146.6250	147.3750	10.2500	10.5625	14.0000	14.1875
7.....	27.5625	27.8750	31.0000	145.6250	146.2500	10.4375	10.7500	14.3125	14.5000
8.....	28.1250	28.4375	32.0000	148.6250	149.2500	10.8750	11.0625	14.5625	14.7500
9.....	28.5000	28.7500	32.2500	148.8750	149.5000	11.1250	11.2500	14.6875	14.7500
10.....	28.5625	28.8750	32.7500	149.1250	149.8750	11.1250	11.2500	14.8750	14.8750
13.....	29.1875	28.4375	33.7500	150.0000	150.6250	11.2500	11.3750	15.1250	15.1250
14.....	28.7500	29.0000	33.7500	149.2500	149.8750	10.8750	11.0000	14.8750	14.8750
15.....	28.3125	28.5625	33.2500	148.8750	149.6250	10.5625	10.7500	14.9375	14.8125
16.....	28.3125	28.5625	32.7500	149.7500	150.5000	10.6875	10.9375	14.9375	14.8750
17.....	28.1250	28.3750	32.5000	149.3750	150.0000	10.5625	10.8125	14.8125	14.8125
20.....	28.0000	28.2500	32.2500	148.6250	149.5000	10.3750	10.6250	14.7500	14.7500
21.....	28.1875	28.4375	32.5000	148.6250	149.5000	10.4375	10.6875	14.6250	14.6250
22.....	28.0625	28.3750	32.5000	148.5000	149.2500	10.5000	10.6250	14.5625	14.5625
23.....	28.2500	28.5625	32.5000	149.2500	150.0000	10.6250	10.8125	14.7500	14.7500
24.....	28.6250	28.8750	32 7500	151.2500	151.8750	10.6875	10.8125	14.8750	14.8750
27.....	28.5625	28.8125	32.7500	151.3750	152.2500	10.6250	10.8125	14.7500	14.7500
28.....	28.1250	28.3750	32.5000	150.1250	150.8750	10.5000	10.6875	14.5625	14.5625
29.....	28.1250	28.3750	32.5000	150.7500	151.6250	10.3125	10.5625	14.6875	14.6250
30.....	28.1250	28.4375	32.2500	150.6250	151.5000	10.4375	10.6250	14.8125	14.6250
31.....	28.2500	28.5625	32.2500	151.1250	152.0000	10.5000	10.6250	14.8750	14.6875
Av. for month	28.179	.....	32.3700	149.120	.....	10.609	10.821	14.647	14.674

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are based on a minimum

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1932	1933	1932	1933	1932	1933
January	29.780	25.400	19.623	16.883	342.515	336.060
February	30.136	26.074	19.573	16.885	345.141	342.114
March	29.810	27.928	18.336	17.588	363.463	343.138
April	28.298	.....	16.923	.....	374.731	.....
May	27.755	.....	16.868	.....	367.370	.....
June	27.466	.....	16.844	.....	364.471	.....
July	26.700	.....	16.930	.....	354.865	.....
August	27.986	.....	18.000	.....	347.449	.....
September	27.870	.....	17.998	.....	347.003	.....
October	27.195	.....	17.813	.....	339.500	.....
November	26.698	.....	18.099	.....	327.458	.....
December	25.010	.....	17.110	.....	327.683	.....
Year	27.892	.....	17.843	.....	350.137	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery — Electrolytic — Domestic		Export	London Spot — Standard		Electrolytic	
	1932	1933	1933	1932	1933	1932	1933
January	7.060	4.775	4.741	39.459	28.557	46.200	33.244
February	5.965	4.775	4.710	36.917	28.481	41.381	32.556
March	5.763	5.011	4.779	33.039	28.179	36.786	32.370
April	5.565	.....	.....	29.943	.....	34.190	.....
May	5.237	.....	.....	28.548	.....	32.833	.....
June	5.145	.....	.....	26.872	.....	30.841	.....
July	5.053	.....	.....	26.071	.....	29.107	.....
August	5.219	.....	.....	31.401	.....	34.784	.....
September	5.978	.....	.....	34.986	.....	38.318	.....
October	5.733	.....	.....	31.890	.....	36.190	.....
November	5.131	.....	.....	31.972	.....	36.568	.....
December	4.813	.....	.....	29.088	.....	34.344	.....
Year	5.555	.....	.....	31.682	.....	35.962	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Lead

	New York		St. Louis		London			
	1932	1933	1932	1933	1932 Spot	1932 3 Mos.	1933 Spot	1933 3 Mos.
January	3.750	3.000	3.550	2.875	15.084	15.128	10.458	10.833
February	3.712	3.000	3.499	2.875	14.560	14.571	10.431	10.719
March	3.150	3.146	2.993	3.021	12.345	12.634	10.609	10.821
April	3.000	.....	2.900	.....	11.223	11.503	.....	.....
May	3.000	.....	2.900	.....	10.673	11.036	.....	.....
June	2.993	.....	2.896	.....	9.608	9.898	.....	.....
July	2.747	.....	2.611	.....	9.818	10.152	.....	.....
August	3.235	.....	3.093	.....	11.349	11.588	.....	.....
September	3.465	.....	3.315	.....	13.122	13.349	.....	.....
October	3.052	.....	2.939	.....	11.958	12.170	.....	.....
November	3.050	.....	2.931	.....	12.071	12.395	.....	.....
December	3.000	.....	2.877	.....	11.144	11.541	.....	.....
Year	3.180	.....	3.042	.....	11.913	12.164	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1932	1933	1932	1933
	Straits		Spot	
January	21.804	22.692	140.219	145.708
February	22.018	23.500	139.143	148.544
March	21.863	24.221	129.810	149.120
April	19.244	.....	108.935	.....
May	20.948	.....	122.286	.....
June	19.659	.....	114.530	.....
July	20.931	.....	125.863	.....
August	22.985	.....	142.017	.....
September	24.779	.....	152.705	.....
October	23.936	.....	151.280	.....
November	23.354	.....	153.585	.....
December	22.677	.....	149.806	.....
Year	22.017	.....	135.848	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1932	1933	1932 Spot	1932 3 Mos.	1933 Spot	1933 3 Mos
January	3.011	3.018	14.416	14.834	14.381	14.595
February	2.817	2.666	13.872	14.289	13.866	14.119
March	2.787	2.987	12.616	13.024	14.647	14.674
April	2.725	.....	11.670	11.958	.....	.....
May	2.532	.....	12.432	12.682	.....	.....
June	2.777	.....	11.548	11.866	.....	.....
July	2.537	.....	11.592	11.967	.....	.....
August	2.758	.....	13.594	13.844	.....	.....
September	3.322	.....	15.455	15.616	.....	.....
October	3.027	.....	14.869	15.140	.....	.....
November	3.094	.....	15.264	15.534	.....	.....
December	3.124	.....	15.209	15.347	.....	.....
Year	2.876	.....	13.545	13.842	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1932	1933	1932	1933
January	55.000	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	.....	23.300	.....
May	55.000	.....	23.300	.....
June	55.000	.....	23.300	.....
July	55.000	.....	23.300	.....
August	55.000	.....	23.300	.....
September	55.000	.....	23.300	.....
October	55.000	.....	23.300	.....
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	55.000	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade.
Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1932	1933	1932	1933	1932	1933
January	5.976	5.722	64.900	48.500	40.000	26.480
February	6.489	5.738	66.304	48.614	40.000	24.000
March	6.188	5.901	72.537	52.676	40.000	24.667
April	5.746	.....	72.125	.....	40.000	.....
May	5.170	.....	66.380	.....	39.500	.....
June	5.034	.....	59.481	.....	37.500	.....
July	5.000	.....	53.580	.....	35.200	.....
August	5.144	.....	47.444	.....	35.000	.....
September	5.606	.....	47.500	.....	33.800	.....
October	5.575	.....	47.600	.....	33.000	.....
November	5.771	.....	48.750	.....	33.000	.....
December	5.400	.....	48.500	.....	30.462	.....
Year	5.592	.....	57.925	.....	36.455	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron

	Bessemer		Basic		No. 2 Foundry	
	1932	1933	1932	1933	1932	1933
January	16.00	14.50	15.00	14.00	15.50	14.50
February	15.68	14.50	14.68	14.00	15.18	14.50
March	15.50	14.50	14.50	14.00	15.00	14.50
April	15.50	.....	14.50	.....	15.00	.....
May	15.50	.....	14.50	.....	15.00	.....
June	14.85	.....	14.35	.....	14.75	.....
July	14.56	.....	14.00	.....	14.50	.....
August	14.50	.....	14.00	.....	14.50	.....
September	14.50	.....	14.00	.....	14.50	.....
October	14.50	.....	14.00	.....	14.50	.....
November	14.50	.....	14.00	.....	14.50	.....
December	14.50	.....	14.00	.....	14.50	.....
Year	15.01	.....	14.29	.....	14.79	.....

Iron in dollars per long ton freight to Pittsburgh, $1.89. F.o.b. Mahoning and Shenango Valley furnaces,

price basis of 4½c. per pound.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft.

All Figures Except Tin Represent Tons of 2,000 Lb.

	1932										1933	
	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Jan.	Feb.
COPPER—North and South America												
Production, refined, daily average	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Domestic shipments	.....	.....	.....	.....	.....	(a) 18,300	(a) 19,700	(a) 25,500	(a) 25,000	(a) 12,000	(a) 17,000	.....
Foreign shipments	.....	.....	.....	.....	.....	(a) 48,000	(a) 53,700	(a) 55,500	(a) 55,000	.....	.....	.....
Stocks, blister and refined	.....	.....	.....	.....	.....	.....	(b) 775,000	(b) 773,000	(b) 770,000	.....	.....	.....
LEAD — United States												
Production, refined, from domestic ore	30,345	23,236	25,902	26,068	15,819	17,118	20,498	21,092	24,465	21,173	24,615	20,033
Production, secondary and foreign	3,989	3,134	2,941	2,033	3,754	6,308	2,315	6,984	2,873	3,624	2,953	2,377
Production, total, daily rate	1,107	879	930	936	631	756	760	905	901	800	889	861
Shipments reported	31,162	26,081	24,258	21,511	19,723	28,248	26,413	29,764	22,838	24,089	19,030	17,349
Stocks, end of month	169,091	169,091	173,929	180,460	180,255	175,426	171,831	170,171	174,629	176,157	184,693	189,751
ZINC — United States												
Production, daily average	726	688	601	547	476	432	434	480	525	596	641	717
Domestic shipments	22,576	18,046	18,087	14,958	12,896	18,069	21,132	18,801	16,000	15,582	15,040	15,280
Export shipments	.....	.....	.....	24	.....	39	20	20	20	20	40	.....
Stocks, end of month	129,451	132,025	132,580	134,032	135,907	131,203	123,056	119,101	121,798	124,705	129,644	134,440
World Production Rate (Daily Average)												
Copper	.....	.....	.....	.....	.....	.....	(b) 2,500	(b) 2,477	(b) 2,480	.....	(b) 2,260	(b) 2,550
Lead	3,598	3,587	3,711	3,497	3,190	3,334	3,323	3,510	3,460	3,350	3,385	3,401
Zinc	2,623	2,528	2,392	2,343	2,224	2,121	2,120	2,255	2,343	2,381	2,556	2,701
Tin (Metric Tons)	340	312	311	261	178	179	255	249	258	239	242	254

(a) Deliveries from all sources; private estimate. (b) *E.&M.J.* estimate.

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